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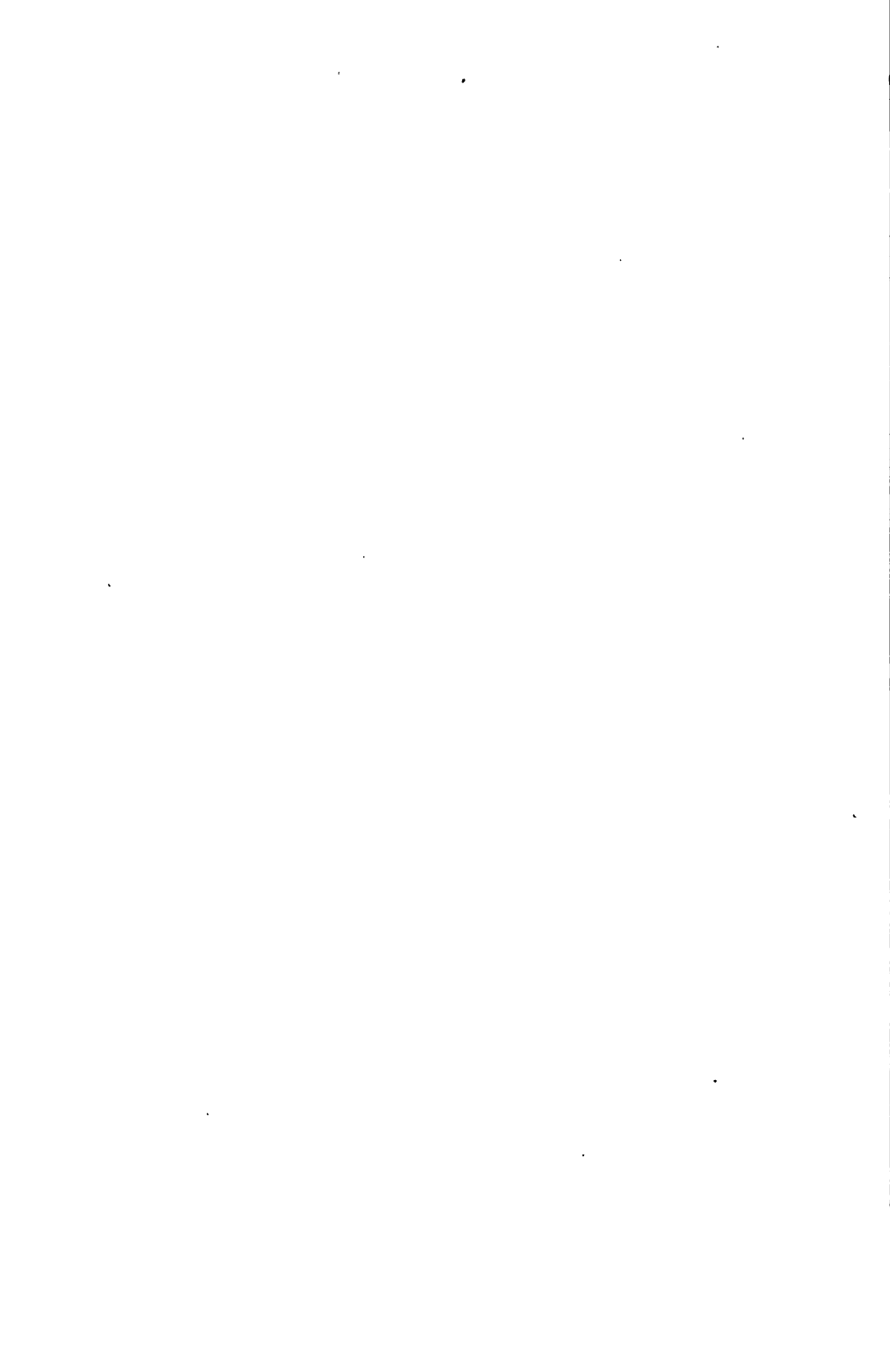
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JANUARY, 1916



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 109

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1916

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY SROUGHTON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

ADVANTAGES OF LIFE MEMBERSHIP

The Institute calls the attention of members to the following advantages of obtaining life membership:

1. One payment avoids the trouble of future payments.
2. The interest on \$150 is less than the annual dues.
3. Provision is thus made early in life for the less productive years.
4. In case of an increase of dues at any time in the future life members are exempt from increased payments.

Excerpt from By-Laws. Article III. Section 3

Any Member or Associate not in arrears may become, by the payment of one hundred and fifty dollars at one time, a Life Member or Life Associate, and shall not be liable thereafter to pay annual dues. The money thus received shall be invested and only the income thereof used for current expenses of the Institute.

No formality is required in taking out a Life Membership except to inclose check for \$150 with a statement that it is intended as a Life Membership payment. In receipt for this an embossed certificate is issued.

JOHN FRITZ MEDAL AWARDED TO DR. JAMES DOUGLAS

An Appreciation of Dr. Douglas by Dr. ALBERT R. LEDOUX

The *Bulletins* of the American Institute of Mining Engineers and the program of the International Engineering Congress, held last September at San Francisco, called attention to an important feature in the proceedings of the Congress; viz., the public presentation to Dr. James Douglas of the John Fritz Medal, for notable achievements in mining, metallurgy, education and industrial welfare. Dr. Douglas' health, unfortunately, did not permit of his taking the journey to California. While he had been willing that the bestowal of the medal should be a public ceremony, his natural modesty made him shrink somewhat from what he considered something of an ordeal, and he begged those who were to take part in it to use restraint in anything which they might have to say as to his work and attainments.

It was, therefore, with some considerable satisfaction on his side that he learned that the medal would be given him without a public ceremony in New York, or elsewhere. On Dec. 5, 1915, the writer of this had the pleasure of handing him the medal, certificate, and official letters of transmittal, at his home at Spuyten Duyvil, where he was surrounded by his children and grandchildren, with one or two of his most intimate friends. He begged the writer to return his thanks to the Committee of Award, and again to express the feeling that nothing in his career merited so great an honor. On this point all of the members of the Institute will share the writer's feeling that Dr. Douglas is too modest.

James Douglas was born at Quebec, Canada, in November, 1837. His father was a distinguished physician and surgeon, employing his skill in the field of philanthropy. He established the first retreat for the insane in the Dominion, to which he devoted himself up to the time of his departure from Canada, when it was taken over by the Government.

The son spent 2 years in study at the University of Edinburgh, which he entered in 1855. Returning to Canada, he graduated from Queens University, at Kingston, Ontario, with the degree of Bachelor of Arts, having also studied medicine, and later theology. After his graduation he traveled extensively with his father in Europe and in the Orient, visiting Egypt several times. They returned with important archæological collections, which Dr. Douglas subsequently donated to the Metropolitan Museum of Art in this city. He then returned to Edinburgh, where he continued his course in medicine, surgery being perhaps his chief interest at that time. He was subsequently licensed to preach and his contemporaries bore testimony to his broad philanthropy and to the sympathy which dominated his every act. His taste was distinctly literary; he had carried off a prize at Edinburgh in English literature.

While still looking toward the ministry or medicine, or a combination of both, as probably his life work, and still occupied with his pen in literary lines, circumstances caused a complete change in his plans. His father had invested heavily in gold and copper mining in Canada,



DR. JAMES DOUGLAS.

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so many ounces. He replied, "No wonder you froze up! Give the furnace so many pounds." The superintendent exclaimed, "Why, if only a few ounces of air blown in freezes the charge, the same result would be hastened if we increased the pressure!" Nevertheless, they apparently tried it after we left, and when we arrived at our hotel in London, Dr. Douglas received a telegram from the works manager, telling him of their success and thanking him for his hint. Afterwards, he received a formal vote of thanks from the Directors.

The mines and railroads controlled by Phelps, Dodge & Co. in the active management of which Dr. Douglas, as President, has been prominent up to the present time, have been very profitable. The most modern devices have been adopted in mills and in smelteries; for instance, he was the first in America to install for the generation of electric power at such plants very large gas engines using wood, and Loomis gas producers. He also was one of the first to introduce in the Southwest the Bessemer converter and the first in the country to employ the trough form.

During all these years of active business life, busy with invention or adaptation of processes; with expansion and consolidation, Dr. Douglas' pen was at work in other fields of thought and activity, and his benefactions were also many. While most of the latter are not known to the public and should not be mentioned here, there is hardly a deserving philanthropic effort in the vicinity of New York or in Eastern Canada that has not been helped and stimulated by him. Among his public benefactions are endowments of colleges, of the Radium Institute in London, the giving of large sums of money in this country to promote the study of cancer, and many others which need not be specified.

He was given the degree of LL. D. both by Queen's University and by McGill University; has been twice President of the American Institute of Mining Engineers. He has been the recipient of the gold medal of the Institution of Mining & Metallurgy.

A list of his writing would be too long for the purposes of this article, but among them may be mentioned:

The Copper Deposits of Harvey Hill, 1870.

Spectroscopic Observations of the Sun, 1870.

The Copper Mines of Chili, 1872.

Copper Mines of Lake Superior, 1874.

Metallurgy of Copper, 1883.

Cupola Smelting of Copper, 1885.

American Methods and Appliances in the Metallurgy of Copper, Lead, Gold and Silver, 1895.

Progress of Metallurgy and Metal Mining in America during the last Half Century, 1897.

Record of Boring in the Sulphur Spring Valley of Arizona, 1898.

Treatment of Copper Mattes in the Bessemer Converter, 1899.

The Characteristics and Conditions of Technical Progress of the 19th Century, 1899.

Gas for use in the Manufacture of Steel, 1902.

Untechnical Addresses on Technical Subjects, 1908.

The Influence of Railroads of the United States and Canada on the Mineral Industry, 1909.

Earthquakes in Mines, 1911.

Development of the Railroads of North America and their Control by the State, 1911.

The Copper Bearing Traps of the Coppermine River, 1913.

Most of the above citations and many others appeared first in the Transactions of various technical and other societies, but in addition to these, Dr. Douglas has given us several historical volumes, among them:

Canadian Independence, Annexation and Imperial Federation.

Old France in the New World.

New England and New France.

Journal and Reminiscences of James Douglas, M. D., by his son.

Enough has been said to show how worthy is the recipient of the distinguished honor conferred upon him through the award of the John Fritz Medal. Personal and intimate association of many years standing and in many fields of activity, have only served to deepen the admiration of the writer for Dr. Douglas, as a man, as a scientist and as a gentleman. Although thinking great thoughts and being associated with great men, nothing was too small to escape his attention; nothing too insignificant to awaken his sympathy. Possibly even in the pages of a publication devoted to technical things, it may not be out of place to say that once when the writer was associated with him in the testing of tin mines in North Carolina, we came across an old prospect shaft, some 10 ft. deep, in the bottom of which he saw a number of frogs or toads which had fallen in and could not escape. Although his time was limited and the work ahead considerable, he insisted upon bringing a fence rail, clambering down, catching the elusive prisoners and tossing them out to safety before he would go on.

NEW YORK MEETING

One Hundred and Twelfth Meeting of the Institute, Monday, February 14, to Thursday, February 17, inclusive, 1916

COMMITTEE ON ARRANGEMENTS

DAVID H. BROWNE, *Chairman.*

BRADLEY STOUGHTON, *Vice-Chairman.*

LAWRENCE ADDICKS,

P. E. BARBOUR,

G. D. BARRON,

J. V. N. DORR,

J. R. FINLAY

L. D. HUNTOON,

BURR A. ROBINSON,

E. M. SHIPP,

JOSEPH STRUTHERS,

E. B. STURGIS,

R. H. VAIL.

Hotel Accommodations.—The Committee on Arrangements is canvassing a number of hotels in New York City, securing wherever possible special rates to members and guests for the days of the meeting. It is possible to secure good accommodations on very reasonable terms in New York City if one knows where to look for them. A list of hotels and rates will be published in the final announcement of the meeting, which will go to the members about the middle of January, in order that those who desire to secure good hotel accommodations at a reasonable price may have the information.

Entertainment and Smoker.—On Tuesday evening, February 15th, an entertainment and smoker will be given. This will be entirely informal and non-technical in character. The Committee in Charge, consisting of

LAWRENCE ADDICKS, *Chairman*

L. D. HUNTOON,

W. B. MCKINLAY,

GILBERT RIGG,

is maintaining a great deal of mystery about the arrangements for this meeting, but promises that those who miss the smoker will regret it for some years to come.

Annual Dinner.—The Annual Dinner will be held at the Hotel Astor on the evening of Wednesday, February 16th, and at this dinner the presence of ladies is especially urged. Tables are arranged for six and eight persons, but larger tables may be secured by application in advance. The dinner will be followed by dancing in the Ball Room of the Hotel Astor.

Entertainment of Ladies.—The Ladies' Committee consisting of

<i>MRS. LOUIS D. HUNTOON, Chairman,</i>	
MRS. LAWRENCE ADDICKS,	MRS. L. O. KELLOGG,
MRS. GEORGE D. BROWN,	MISS ELIZABETH H. KUNZ,
MRS. DAVID H. BROWNE,	MRS. A. R. LEDOUX,
MRS. J. V. N. DORE,	MRS. WILLARD S. MORSE,
MRS. ARTHUR S. DWIGHT,	MRS. HENRY S. MUNROE,
MRS. KARL EILERS,	MRS. H. A. PROSSER,
MRS. H. W. HARDINGE,	MRS. CHARLES F. RAND,
MRS. LEVI HOLBROOK,	MRS. THOMAS ROBINS,
MRS. AXEL O. IHLSENG,	MRS. BURE A. ROBINSON,
MRS. W. R. INGALLS,	MRS. E. M. SHIPP,
MRS. J. H. JANEWAY,	MRS. BRADLEY STOUTON,
MRS. SIDNEY J. JENNINGS,	MRS. LYMAN B. STURGIS,
<i>MRS. B. B. THAYER,</i>	

has already made plans for the entertainment of the ladies. The Ladies' Committee will be in the Reception Room on the first floor of the Engineering Societies Building from 12 to 1 o'clock each day to receive visitors and escort them to the luncheons. The presence of ladies at the luncheons and the Annual Dinner is especially requested. Arrangements have been made for the following functions for the entertainment of ladies:

Monday noon: Luncheon at Engineering Societies' Building.

Monday afternoon: Thé Dansant at one of the New York hotels.

Tuesday noon: Luncheon at Engineering Societies' Building.

Tuesday afternoon: Hippodrome to see fancy ice-skating.

Wednesday noon: Luncheon at Engineering Societies' Building.

Wednesday afternoon: Visit to the Art Galleries of Senator William A. Clark, followed by tea at the nearby residence of a member of the Ladies Committee.

Wednesday evening: Annual dinner followed by dancing.

Thursday: All-day boat excursion, which see.

Automobiles will be furnished, in connection with all these entertainments, for the use of ladies, and it is earnestly desired that not only visiting ladies, but the wives of the local members, will attend the luncheons and the entertainments afterwards. It is also especially urged that the ladies attend the all-day boat trip around New York Harbor, by courtesy of the Secretary of the Navy. This trip is described more fully in the following paragraph.

All-day Boat Excursion.—By courtesy of the Secretary of the Navy a Naval vessel will be furnished for the members and guests of the Institute for a cruise in New York Harbor. Landings will be made at some point in the Harbor and by courtesy of the Secretary of War the party will be allowed to witness the firing of some of the large guns.

Gathering of College Alumni.—Monday evening, February 14th, has

been designated as "College Night" and at this time it is expected that formal gatherings of many of the college alumni will be held. A Committee on College Reunions, of which Dr. Joseph Struthers is Chairman, has charge of this part of the activities of the meeting. Sub-Committees have been formed for the following colleges and associations:

University of Wisconsin,	Washington State College,
University of Missouri,	Lehigh University,
University of Washington,	Case School of Applied Science,
University of Idaho,	Columbia University,
University of Kansas,	Harvard University,
Lafayette College,	Yale University,
Massachusetts Inst. of Technology,	Rensselaer Polytechnic Institute,
Pennsylvania State College,	Old Freibergers,
Colorado School of Mines,	Michigan College of Mines,
Tufts College,	University of Pennsylvania,
	University of Pittsburgh.

Opportunities will be given to all members and guests to register their college affiliations, so that the alumni of the different institutions can be informed of the presence of their colleagues at the meeting. Several of the colleges on the list given above have clubs in New York City and to those colleges which have no headquarters in the City the privileges of the Engineers' Club or the Chemists' Club for dinner or other informal entertainments may be obtained. The same privileges will be extended to individual visitors from outside the city. On "College Night" the rooms of the Institute will be kept open for the benefit of any members who do not have any other place of meeting, and especially for those who are not affiliated with any college. Rooms will also be available here for meetings of college alumni if desired. For Columbia graduates an entertainment will be given at the Columbia Club on this evening, the details of which will be announced later.

Luncheons.—Luncheons will be served each day on the fifth floor of the Engineering Societies Building between the morning and afternoon technical sessions, and luncheon will also be served on board the boat during the all-day trip referred to in the previous paragraph.

Registration Facilities.—Registration facilities will be available during each day of the meeting. The Registration Bureau will be on the fifth floor where the technical sessions and luncheons will be held.

New York Club Facilities Offered.—The following New York Clubs courteously offered their privileges to visiting members and guests during the period of the meeting last year and it is hoped that similar courtesies may be extended this year:

Chemists' Club, 52 E. 41st St.
Columbia University Club, 18 Gramercy Park
Harvard Club, 27 West 44th St.
Machinery Club, 50 Church St.
Princeton Club, 121 E. 21st St.
Rocky Mountain Club, 65 W. 44th St.
Technology Club, 17 Gramercy Park
Williams Club, 291 Madison Ave.
Yale Club, Vanderbilt Ave. and 44th St.

Facilities for Members at Institute Headquarters.—The Institute maintains for the use of members (and especially for the use of out-of-town members) a reading and writing room, where all usual office facilities are available, including telephone, telegraph, receipt, forwarding and care of mail and express packages, etc. The Institute also maintains

for the use of its members free dressing rooms, library research department, etc. Attention is called to these facilities at the present time in the belief that they may be especially useful to out-of-town members in connection with the Annual Meeting.

Library Service.—Special efforts are made to make the Library as serviceable as possible to members who are distant from headquarters and this is accomplished practically by the means of searches, copying and translation facilities, etc. More detail on this matter is given in the section of the Bulletin devoted to the library.

Members' Writing Room.—At the headquarters of the Institute, a Members' Writing Room is maintained, equipped with writing, telephone, telegraph, messenger, dictation and other office facilities, including a pay-station typewriter. A file of current magazines is also maintained in this room and the Institute's staff will lend its best efforts to serving the members in connection therewith.

Dressing Rooms.—On the ground floor of the Engineering Societies Building, are a number of free dressing rooms, where members will find the necessary facilities. This is particularly intended for members who arrive in New York on early trains, or who desire to make a change of clothes for evening entertainments. The building is open from 8 a.m. until 10 p.m. and access may be had at other hours by ringing the door-bell.

Technical Program.—The following papers have been received and accepted by the Committee on Papers and Publications:

Precious and Base Metals

The Behavior of Stibnite in an Oxidizing Roast. By H. O. Hofman and John Blatchford.

Determination of Antimony in the Products Obtained by Roasting Stibnite. By W. T. Hall and John Blatchford.

Recent Advances in the Chemistry of the Cyanogen Compounds. By J. E. Clennell. The Newnam Hearth. By W. E. Newnam.

A Development of Practical Substitutes for Platinum and its Alloys, with Special Reference to Alloys of Tungsten and Molybdenum. By F. A. Fahrenwald.

Non-metallic Minerals

Fire Clay. By L. C. Morganroth.

Mining Methods

Broken Hill Underground Mining Methods. By E. J. Horwood.

Underground Mining Methods of Utah Copper Co. By Thomas S. Carnahan.

Milling Methods

Notes on Flotation. By John M. Callow.

Electricity in Mines

Tests on Motor-Driven Equipment for Use in Preparation of Anthracite Coal. By H. M. Warren, A. S. Biesecker and E. J. Powell.

An Electro-Hydraulic Shovel. By Frank H. Armstrong.

Conservation

Conservation of Iron Ores. By C. K. Leith.

Conservation and Economic Theory. By R. T. Ely.

Petroleum and Gas

Development of the Law Relating to the Use of Gas Compressors in Natural Gas Production. By Samuel S. Wyer.

The Control of Petroleum and Natural Gas Wells. By Alfred G. Heggem.

The Revolution of Drilling Rigs. By R. B. Woodworth.

Necessary Use and Effect of Gas Compressors on Natural-Gas Field Operating Conditions. By Samuel S. Wyer.

Iron and Steel

- The Control of Chill in Cast Iron. By G. M. Thrasher.
 Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains. By R. J. Wysor.
 Washed Metal. By H. D. Hibbard and E. L. Ford.
 Manganese-Steel Castings in the Mining Industry. By Walter S. McKee.
 Magnetic Studies of Mechanical Deformation in Certain Ferromagnetic Metals and Alloys. By H. Hanemann and P. D. Merica.
 Effect of Carbon on the Physical Properties of Heat-Treated Steel. By J. H. Nead.
 The Determination of Grain Size in Metals. By Zay Jeffries, A. H. Kline, and E. B. Zimmer.
 Metallography of Steel for U. S. Naval Ordnance. By H. E. Cook.
 A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron. By W. McA. Johnson.
 Trials of Ferrotitanium in Acid Open-Hearth Steel Castings. By Edwin F. Cone.
 The Iron Mines of the Sierra Menera District of Spain. By Victor de Ysassi.

Geology

- The Iron Deposits of Daiquiri, Cuba. By Waldemar Lindgren and C. P. Ross.
 The Disseminated Copper Ores of Bingham Canyon, Utah. By J. J. Beeson.
 Geology of the Ore Deposits of the Tintic District. By G. W. Crane.

Coal and Coke

- Illumination of Mines. By Robert P. Burrows.
 Some Researches on Fire-Damp. By E. Hauser.
 Economies in a Small Coal Mine. By H. A. Everest.
 The Effect of Aeration and "Watering Out" on the Sulphur Content of Coke. By J. R. Campbell.

Brass

- Recrystallization of Cold-Worked Alpha Brass on Annealing. By C. H. Mathewson and Arthur Phillips.

Most of them have already been published in the *Bulletin*. Other papers are in the hands of the Committee so that there will no doubt be additions to this list before the meeting opens and all papers will be published in the *Bulletin* in advance of the meeting.

Members desiring to communicate directly with members of the Committee on Arrangements or Chairman of a special committee may address them at Institute Headquarters and the letters will be promptly forwarded.

ENGINEERS' RESERVE CORPS

D. M. Riordan has been appointed a member of the Institute Committee of the Engineers' Reserve Corps movement. Our Committee, therefore, now consists of Henry S. Drinker, Chairman, Messrs. Arthur S. Dwight, Warren A. Wilbur and D. M. Riordan.

MILITARY READING FOR CIVILIAN ENGINEERS

By authority of the Secretary of War, and in response to frequent requests, the following suggested list of reading is issued by the War Department for the information of civilian engineers desiring to inform themselves on military subjects:

These references have been selected, first, with a view to giving to engineers unfamiliar with the art of war a general survey of that sub-

ject, an understanding of which is the first essential to insure successful application of engineering knowledge and resources to military purposes; and, second, with a view to setting forth, as far as practicable, the ways in which engineering is applied to military purposes and the means provided therefor.

Both military art and military engineering are progressive, and a considerable part of the latest and most detailed information published is available only in service journals of our own and foreign armies. This is particularly true of technical details of seacoast defense (including submarine mining), of field artillery, of military aviation, and the influence of these on military engineering. It is believed, however, that the fundamentals of each subject are well covered by the references given in this list. While the list is long, the relative importance of the various works is indicated, and suitable comments on each are included, so that persons using the lists of references may be able to select those which particularly interest them.

The references under each subject are generally divided into two groups, the first containing the more essential references, and the second those suitable for persons desiring to inquire further into the subject.

Suggestions looking to improvements of the lists will be gladly received.

NOTE.—The following abbreviations are used: Supt. of Docs.: Superintendent of Documents, Government Printing Office, Washington, D. C.

Book Dept.: Book Department, Army Service Schools, Fort Leavenworth, Kan.

A.—MILITARY POLICY, CONDUCT OF WAR, AND MILITARY HISTORY

Group I

1. OFFICIAL BULLETIN, vol. i, No. 2, Office of the Chief of Staff, Washington, D. C. (Especially pp. 21–39.) Publisher: Army War College, Washington, D. C. Free. (An official outline of the theory under which our forces are to be organized and administered.)
2. MILITARY POLICY OF THE UNITED STATES. Upton. May be obtained from Supt. of Docs.; paper, 50 cents; cloth, 65 cents. (A most valuable and comprehensive review of this subject.)
3. FIELD SERVICE REGULATIONS, 1914. May be obtained from Supt. of Docs.; 60 cents. (A condensed official statement of principles, methods, and details of military operations.)
4. ELEMENTS OF STRATEGY. Fiebeger. Publisher: U. S. Military Academy, West Point, N. Y. May be obtained from Book Dept.; 75 cents. (A short outline, with historical illustrations.)

Group II

5. CONDUCT OF WAR. Von der Goltz; translated by J. Dickman, Hudson Publishing Co., Kansas City, Mo. May be obtained from Book Dept.; \$1.70. (The standard work on this subject, covering generally the same ground as 4, but more abstractly and elaborately.)
6. ON WAR. Clausewitz; translated by J. J. Graham; 3 vols.; K. Paul, Trench, Trubner and Co., 1908. May be obtained from Book Dept.; \$6.60 (including postage and duty). (The greatest classic on the subject; a complete analysis of the phenomenon of war, and profound discussion of the mechanism thereof. Written early in the 19th Century, it is still the foundation of modern military theory.)
- 6½. THE NATION IN ARMS. Von der Goltz. May be obtained from Book Dept.; \$2.50. (An excellent modern work on war; less elaborate but more readable than Clausewitz.)

7. **AMERICAN CAMPAIGNS.** M. F. Steele; 2 vols.; Publishers: Byron S. Adams Publishing Co., Washington, D. C. May be obtained from Book Dept.; \$4.50. (In addition to careful historical surveys of all the campaigns from the Colonial Wars to the Spanish American war, these lectures give extensive and valuable comments as to the military principles.)
8. **A STUDY OF ATTACKS ON FORTIFIED HARBORS.** Rodgers; Proceedings Nos. 111, 112, and 113, U. S. Naval Institute, Annapolis, Md.
9. **LESSONS OF THE WAR WITH SPAIN.** Mahan. Publishers: Little, Brown & Co., Boston, Mass. May be obtained from Book Dept.; \$2. (Of special importance, as showing the true relation between our coast defense and our navy.)
10. **REPORTS OF MILITARY OBSERVERS ON THE RUSSO-JAPANESE WAR.** Part III, J. E. Kuhn. May be obtained from Supt. of Docs.; 60 cents. (In addition to an account of operations, this report contains valuable information as to fortification and siege work, organization, and equipment.)
11. **ORGANIZATION AND OPERATION OF THE LINES OF COMMUNICATIONS IN WAR.** Furse, 1894; Publishers: Wm. Clowes & Sons, Ltd., London. (An old but comprehensive survey of this subject, with much historical information.)

B.—PERMANENT FORTIFICATIONS

Group I

- (The references given cover chiefly the principles and general features of this subject; the *details* are mostly printed in unavailable form, either in service journals or in confidential documents. References to some of the former can be furnished, if desired.)
12. **REPORT OF NATIONAL COAST DEFENSE.** (Taft) Board, 1906. May be obtained from Army War College, Washington, D. C. Free. (The official project for harbor defenses of the United States. On account of progressive obsolescence of seacoast defenses, this project has been, or is being, modified, but still sets forth clearly the fundamentals of its subject.)

Group II

13. **LECTURES ON SEACOAST DEFENSE.** Winslow. Publishers: U. S. Engineer School, Washington Barracks, D. C. Price 50 cents. (Much of these lectures relates to technical details, and a considerable part is now obsolete.)
14. **PERMANENT FORTIFICATIONS.** Fieberger, 1900; U. S. Military Academy, West Point, N. Y.; \$1. May be obtained from Book Dept. (While rather old, this work gives a simple presentation of the fundamentals on its subject, including an historical outline. A revised edition will soon be published.)
15. **FORTIFICATIONS.** C. S. Clarke. Dutton & Co., New York; \$4.50. May be obtained from Book Dept. (A treatise on the same lines as 14.)
16. **PRINCIPLES OF LAND DEFENSE.** Thuillier, 1902; Longmans, Green & Co. May be obtained from Book Dept.; \$3.83. (A valuable work, covering the principles of both field and permanent fortification.)

C.—ORGANIZATION, EQUIPMENT, AND DUTIES OF ENGINEER TROOPS

Group I

17. **FIELD SERVICE REGULATIONS, 1914.** (See A. 3.)
18. **TABLES OF ORGANIZATION, 1914.** May be obtained from Supt. of Docs.; 25 cents. (These tables represent—subject to modification and within the limits of existing law—the approved policy of the War Department with regards to organization.)
19. **OFFICIAL BULLETIN, Office of the Chief of Staff, vol. i, No. 4 (Appendix 4).** Use of Engineer Troops. Publisher: Army War College, Washington, D. C. Free. (An official statement of the principles which should govern in the use of engineers, with practical suggestions.)
20. **DUTIES OF ENGINEER TROOPS IN A GENERAL ENGAGEMENT OF A MIXED FORCE.** Burgess. Publisher: U. S. Engineer School, Washington Barracks, D. C.; 25 cents. (Obsolete in some respects, particularly organization, but excellent in general scope.)
21. **GENERAL ORDERS No. 6, WAR DEPARTMENT, 1915.** May be obtained from the Adjutant General, U. S. Army, Washington, D. C. Free. (Prescribes the training of Engineer troops.)

Group II

22. **STUDIES IN MINOR TACTICS.** Army Service Schools, 1915. May be obtained from Book Dept.; 50 cents. (The principles of Minor Tactics are set forth by solution of a series of problems.)
23. **TECHNIQUE OF MODERN TACTICS.** Bond & McDonough, 1914; Banta Publishing Co., Menasha, Wis. May be obtained from Book Dept.; \$2.55. (This work covers, in a very specific way, the principles of tactics for all arms, a general knowledge of which is essential for engineers.)
24. **OPERATION ORDERS.** Von Kiesling; translation. May be obtained from Book Dept.; 50 cents. (A lucid exposition, by use of assumed cases, of the operation of highly trained troops of all arms in various phases of battle.)
25. **ENGINEER UNIT ACCOUNTABILITY MANUAL.** May be obtained from Supt. of Docs.; 5 cents. (Official lists of standard equipment supplied to Engineer battalions and companies.)
26. **ORGANIZATION OF THE BRIDGE EQUIPAGE OF THE U. S. ARMY,** 1915. (Revised edition just going to press.) (Includes description of equipage and regulations for ponton drill.)
27. **OFFICERS' MANUAL.** Moss; Banta Publishing Co., Menasha, Wis.; \$2.50. May be obtained from Book Dept. (Treats of routine duties of officers, customs of the service, army organization, etc.)
28. **MANUAL FOR COURTS MARTIAL.** May be obtained from Supt. of Docs.; 50 cents.

D.—FIELD ENGINEERING

- (Military field engineering at the front differs from ordinary engineering work in the field, it being generally simpler, of a rough-and-ready character, and especially because of the limited equipment which can be taken along with the advance of an army, and because of the necessity of working in strict subordination to the military situation. In rear of the army on the contrary, conditions are very similar to those governing ordinary engineering operations, and civilian organization is suitable, subject to directions by the higher military staff. Little attempt is made in works on military field engineering to treat of general engineering methods.)
29. **FIELD FORTIFICATION.** Fiebeger, 1913; John Wiley & Sons, New York. May be obtained from Book Dept.; \$1.90. (In addition to technical details, this work gives valuable historical illustrations of the principles of this subject.)
 30. **FIELD ENTRENCHMENTS, SPARE WORK FOR RIFLEMEN.** John Murray, London. May be obtained from Book Dept.; 40 cents. (A very up-to-date little work, especially on details.)
 31. **NOTES ON FIELD FORTIFICATION.** Army Field Engineer School. May be obtained from Book Dept.; 30 cents.
 32. **ENGINEER FIELD MANUAL.** Professional Paper No. 29, Corps of Engineers, U. S. Army; 3d edition, 1909. 500 pages. May be obtained from Supt. of Docs.; \$1. (A very complete official pocketbook for Engineer officers in the field, containing much tabular and technical data, as well as brief outlines of principles and methods. The subjects covered are, Part I, Reconnaissance; Part II, Bridges; Part III, Roads; Part IV, Railroads; Part V, Field Fortification, and Part VI, Animal Transportation. A new revision of the manual is contemplated, but will not be ready within a year. The portion of the manual relating to Field Fortification, being somewhat obsolete, should be considered in connection with either 30 and 31 above. The portion relating to Railroads is largely superseded by 35 below.)
 33. **NOTES ON BRIDGES AND BRIDGING.** Spalding. May be obtained from Book Dept. (A small pamphlet on military bridging.)
 34. **MILITARY TOPOGRAPHY FOR MOBILE FORCES.** Sherrill, 2d edition; Banta Publishing Co., Menasha, Wis., 1911. May be obtained from Book Dept.; \$2.25. (Besides matter given in ordinary text-books on surveying, this work gives in detail the special methods of sketching developed in the army for rapid military mapping.)
 35. **MILITARY RAILROADS.** Connor; Professional Paper No. 32, Corps of Engineers, U. S. Army. Supt. of Docs.; 50 cents. (Intended to cover general administration of existing railroads for military purposes and the handling of railroads by military personnel in the advance sections where railroads cannot be operated by their regular civilian organizations, or where new railroads are required in the immediate vicinity of the Army. Revised edition soon to appear.)

36. **NOTES ON MILITARY EXPLOSIVES.** Weaver; J. Wiley & Sons, New York; 1912. May be obtained from Book Dept.; \$2.20. (Elementary notes on this subject will be found in the Engineer Field Manual and other references cited. This work is more elaborate.)

E.—MISCELLANEOUS

37. **REGULATIONS FOR THE ARMY OF THE UNITED STATES.** Supt. of Docs.; 50 cents.
 38. **THE "VOLUNTEER LAW,"** approved April 25, 1914; Bulletin No. 17. War Department, 1914. May be obtained from The Adjutant General, U. S. Army, Washington, D. C. Free.
 39. **GENERAL ORDERS No. 54,** War Department, 1914. May be obtained from The Adjutant General, U. S. Army, Washington, D. C. Free. (Covers examination of candidates for commissions as officers of *volunteers*.)
 40. **GENERAL ORDERS No. 50,** War Department, 1915. May be obtained from The Adjutant General, U. S. Army, Washington, D. C. Free. (Amends General Orders 54, 1914, as to examination of candidates for commissions in volunteer *engineers*.)
 41. **TREATISE ON MILITARY LAW.** Davis; J. Wiley & Sons, New York. May be obtained from Book Dept.; \$5.30.
 42. **ELEMENTS OF MILITARY HYGIENE.** Ashburne; new edition; Houghton, Mifflin & Co., Boston, 1915. May be obtained from Book Dept.; \$1.30.

F. PERIODICALS

43. **PROFESSIONAL MEMOIRS.** Corps of Engineers, U. S. A., and Engineer Department at Large; Bi-monthly (formerly quarterly); Washington Barracks, D. C., Engineer Press; per year, \$3.
 44. **THE ROYAL ENGINEERS' JOURNAL.** Royal Engineers' Institute, Chatham, England; Monthly; per year \$4. (American agents; E. Steiger & Co., 49 Murray St., New York.)
 45. **JOURNAL OF THE MILITARY SERVICE INSTITUTION,** Governors Island, New York. Bi-monthly; published by the Institution; per year \$3.
 46. **JOURNAL OF THE UNITED STATES ARTILLERY.** Bi-monthly; Fort Monroe, Va., Coast Artillery School press; per year \$2.75, including Index to Current Literature; without Index, \$2.50.
 47. **JOURNAL OF THE UNITED STATES CAVALRY ASSOCIATION.** Published by the Association at Fort Leavenworth, Kans.; per year \$2.50.
 48. **INFANTRY JOURNAL.** Bi-monthly; published by the U. S. Infantry Association, Union Trust Building, Washington, D. C.; per year \$3.
 49. **FIELD ARTILLERY JOURNAL,** quarterly; published by the U. S. Field Artillery Association, 601 Star Building, Washington D. C.; per year \$3.

MAGAZINES, ETC., FOR SALE TO COMPLETE MEMBERS' SETS

The Institute has prepared a list of Societies' publications, magazines, etc., which were duplicates and were discarded at the time of the consolidation of the libraries of the three Founder Societies. These books will be disposed of about March 1, 1916 by the Board of Directors of the Institute through the Library Committee. If, meanwhile, members who have incomplete sets of these books, or for any other reason desire to secure some of them, they are asked to communicate immediately with the Secretary, as this is an unusual opportunity to secure books at a very low price.

American Gas Light Journal, vols. 31 to 39.

American Institute of Electrical Engineers, Transactions, vols. 5 to 26.

American Machinist, vols. 3 to 20, 22 to 29.

American Society of Civil Engineers, Transactions, vols. 2 to 10, 21 to 24, 26 to 33, 39 to 49, 52, 55, 58, 74. Index, vols. 1 to 21.

American Society of Mechanical Engineers, Transactions, vols. 1 to 28. Index, vols. 1 to 20, vols. 1 to 25.

- American Waterworks Association, *Proceedings*, 27 to 30.
Annalen für Gewerbe und Bauwesen, vols. 42 to 61.
 Association of Engineering Societies, *Journal*, vols. 20 to 38.
 Canadian Society of Civil Engineers, *Transactions*, vols. 1 to 18.
Cassier's Magazine, vols. 6 to 7, 12 to 32.
Chemical News, vols. 7 to 20, 22.
Dingler's Polytechnisches Journal, vols. 55 to 58, 73 to 74, 95 to 98, 103 to 122, 127 to 174, 176 to 310, 312 to 314, 250 to 264, 266 to 270. Index 1 to 78, 119 to 198.
Engineer (London), vols. 40 to 104.
Engineering, vols. 2 to 84.
Engineering Magazine, vols. 1 to 34.
Engineering and Mining Journal, vols. 43 to 44, 49 to 57, 71 to 74, 78 to 80; vols. 12 to 73; vols. 5 to 11, 15 to 38, 42 to 47, 49 to 53, 55 to 57, 59, 69, 71.
Engineering Record, vols. 24 to 27, 29 to 40, 46 to 54.
 Engineers' Club of Philadelphia, *Proceedings*, vols. 15 to 23.
Fielden's Magazine and Engineering Review, vols. 8 to 16.
 Franklin Institute, *Journal*, vols. 1 to 164.
 Institution of Electrical Engineers, *Journal*, vols. 31 to 38. Index, vols. 1 to 30.
 Institution of Engineers and Shipbuilders in Scotland, *Transactions*, vols. 18, 44 to 50.
 Institution of Mechanical Engineers, *Proceedings*, 1879-1906. General index, 1874-1884.
 Institution of Naval Architects, *Transactions*, vols. 28 to 30; vols. 35 to 50. Index, vols. 1 to 42; vols. 1 to 46.
 Iron and Steel Institute, *Journal*, 1871-1909. Special volume, 1890. Index, 1869-1889, 2 vols.
Journal of the U. S. Artillery, vols. 1 to 28.
 Junior Engineering Society, *Record of Transactions*, vols. 2 to 17.
 Liverpool Engineering Society, *Transactions*, vols. 1 to 18, 22 to 29.
 Master Car Builders' Association, *Report of Proceedings*, vols. 16, 18, 21, 24, 26, 29, 36 to 37, 39, 41, 43, 45, 48.
Metal Worker, vols. 59 to 60, 65, 68 to 78.
Page's Weekly, vols. 2 to 10.
Popular Science Monthly, vols. 1 to 10, 15 to 18, 27 to 55, 57 to 62, 69 to 73.
Railroad and Engineering Journal, vols. 61 to 69, 71 to 80.
Railroad Gazette, vols. 12 to 22, 24 to 32.
Railway Review (Chicago), vols. 36 to 47.
 Royal Society of London, *Philosophical Transactions*, 1792-1812, 1821-1876.
Science, vols. 1 to 9; New series, vols. 14 to 26.
 Science Abstracts, vols. 1 to 10.
 Society of Arts, *Journal*, vols. 28, 31, 33 to 54.
 Society of Chemical Industry, *Journal*, vols. 9, 12 to 12, 14 to 16, 18 to 19, 21 to 26.
 Société des Ingenieurs Civils de France, *Mémoires*, 1875-1906.
Stahl und Eisen, vols. 4 to 9, 11 to 14.
 U. S. Naval Institute, *Proceedings*, vols. 5 to 32. Index, 1 to 15.
Verein deutscher Ingenieure Zeitschrift, vols. 18, 20, 22 to 50. Index, 1884-1903.
 Western Railway Club, *Official Proceedings*, vols. 9 to 19.

SECOND PAN-AMERICAN SCIENTIFIC CONGRESS

At the Second Pan-American Scientific Congress, meeting in Washington, D. C., from Dec. 27, 1915 to Jan. 8, 1916, the following special topics will be discussed in each of the four sub-sections of Section VII, of which Hennen Jennings is Chairman:

A. *Mining*.—The mining law of each country and the changes that may be made to aid the development of mineral resources. History of the mining industry in each country with reference to the beginnings of that industry. The development of the Patio process. Bibliography of mining.

B. *Metallurgy*.—Present methods of concentrating ores and the development of concentration methods. International relations in the exchange of ores and metals. Bibliography of metallurgy.

A. and B. *Mining and Metallurgy*.—Development of hydro-electric power for mining and metallurgy, the amount probably available, and specific benefits from its utilization.

C. *Economic Geology.*—The relation of geological work to the development of the country. A bibliography of economic geology.

D. *Applied Chemistry.*—Natural and artificial nitrates; the present status and the outlook for these industries.

The official hotel headquarters of the Congress will be at the New Willard Hotel, where members may register and receive their assignments to sections. Raleigh Hotel has been designated as the headquarters of Section VII.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Nov. 10 to Dec. 10, 1915:

F. Lynwood Garrison, Philadelphia, Pa.

William J. Lakeland, Burma, India.

Dwight E. Woodbridge, Duluth, Minn.

Chester A. Fulton, Yonkers, N. Y.

Willet G. Miller, Toronto, Canada.

L. G. Huntley, Pittsburgh, Pa.

L. E. Howard, Lockport, N. Y.

H. W. DuBois, Philadelphia, Pa.

Howard R. Stewart, London, England.

E. Fleming P'engle, Costa Rico, C. A.

Fred Maccoy, El Oro, Mexico.

O. N. Scott, Toronto, Ont., Canada.

K. P. Swenson, Tokyo, Japan.

H. L. Mead, New York, N. Y.

E. DeGolyer, Norman, Okla.

J. S. Lane, New York, N. Y.

T. V. Nonactre, Argentina.

N. H. Darton, of the U. S. Geological Survey, has returned to Washington after a long season of field work in New Mexico investigating the prospects of finding potash deposits in the Red beds.

Pierre Bouery, for 15 years manager of LaGrange hydraulic mine at Weaverville, Cal., has been appointed manager of the Valdez Creek placer mines in Alaska.

George B. Holderer has been made general manager of the Furlough Development Co., which is developing a copper property near Wickensburg, Ariz.

William C. Russell has taken the management of the Carriman Mining Co., operating the Caribou property in Boulder County, Col.

Dr. John A. Mathews, for many years manager of the Halcombe Steel Co., of Syracuse, N. Y., has been elected president of this company and the Syracuse Crucible Steel Co.

W. G. Norrie has been appointed manager of the Silver Standard Mine at New Hazelton, B. C., Canada.

Dr. L. D. Ricketts, on account of his work in promoting the mining industry of Arizona, was selected by Governor Hunt to visit the San Francisco Exposition and receive an official greeting as the State's most distinguished citizen.

Edwin Higgins, engineer for the Bureau of Mines in the Lake Superior district, has been transferred to California, where he will take charge of the work for the Bureau, which coöperates with the State in mine inspection work.

Edward W. Maynard, formerly in charge of the Atlas Powder Co. plant at Senter has been made superintendent of the plant near San Francisco, Cal.

Walter O. Snelling, formerly of Pittsburgh, has purchased property in Long Island City, N. Y., on which he proposes to build a large laboratory for chemical research.

Dr. Charles R. Van Hise is chairman of a commission appointed to investigate the slides in the Panama Canal and to determine the probable causes.

T. Skewes Saunders has been appointed General Manager of the Dos Estrellas Mining Co., El Oro, Mexico, succeeding **André P. Griffiths**.

H. W. Hardinge has been awarded the John Scott Medal and the maximum premium by the City of Philadelphia upon the recommendation of the Franklin Institute, for the invention of his conical mill. The merits of the mill have been under investigation of the engineering staff of the Franklin Institute for over two years. From a bulletin published by the Franklin Institute, of Philadelphia, the following facts about the John Scott Medal are obtainable.

"John Scott, of Edinburgh, Scotland, in the year 1816 bequeathed, in trust, to the City of Philadelphia the sum of \$4,000, directing that the interest and dividend, to become receivable therefrom be paid out in premiums, to be distributed among ingenious men and women who make useful inventions, but no such premium to exceed \$20, and therewith should be given a copper medal with this inscription 'To the most deserving.'"

The City of Philadelphia in February, 1834, vested the awarding of the aforesaid premiums and medals to the Franklin Institute, and by this act the Franklin Institute, through its scientific body, represents the City of Philadelphia."

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Two metallurgical chemists for permanent positions in Siberia; one man to have charge of laboratory dealing with the products of a mill working silver-lead-zinc ores, the other to have charge of laboratory in connection with smelting these products. Established camp; living conditions and climate good. Married men preferred. No. 57.

An electro-metallurgist for small zinc plant now under construction. Moderate salary to start but likelihood of substantial increase if plant proves successful. No. 58.

Assistant engineer in office of consulting engineer for large mining and smelting company. Man desired with technical education and broad experience in mechanical line in connection with mining and smelting operations, capable of taking charge of office and drafting room. No. 59.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, aged 34, married, technical graduate, 10 years' experience as mining engineer and mine superintendent, desires position. Can show low cost records and good references. No. 176.

Member, mechanical superintendent, wants position. Has had 12 years' experience as master mechanic of smelting plants in United States and Mexico. Also superintendent of shops doing marine, mining, and general contracting work. Experienced in manufacture of mining tools equipment. Up to the minute in copper, lead, and zinc smelting. Has learned a great deal, can forget much, and has the inclination and ability to learn more. Best of references. No. 255.

Member, graduate of Columbia School of Mines; aged 45, with experience as mine surveyor, engineer, and superintendent, and as designing, manufacturing and constructing engineer; familiar with mining methods, with handling of men, with machinery, with designing, with erection and construction work. Available for long or short engagements in United States or Canada. Now in the West. No. 263.

Graduate of Michigan College of Mines, with 15 years' experience in engineering, exploration, equipment, efficiency and safety work, management and executive. Experience obtained in Lake Superior Region, some foreign experience. Good record and references. No. 272.

Recent graduate, woman, having knowledge of analytical and metallurgical chemistry, and courses in assaying, mineralogy and geology, desires position as chemist or assistant chemist. No. 273.

Member, graduate of Massachusetts Institute of Technology, with 10 years' experience in geology related to mining, 5 years in the Northeastern and 5 in the Northwestern States. Especially familiar with geology of iron, copper, lead-silver and gold. Qualified in microscopic determinations. Desires position as field engineer or with mining company. At present in government employ. No. 274.

Young technical graduate, S. B. and S. M., speaks Spanish, experience in Arizona and Mexico, desires geological work of any kind, go anywhere. No. 275.

NEW YORK SECTION

Executive Committee

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P. A. MOSMAN, *Treasurer*

LEWIS W. FRANCIS

BENJAMIN B. LAWRENCE

At the meeting of the *New York Section* held at the Machinery Club on Dec. 8, 1915, the subject was "The Engineer in Modern Warfare, with Especial Reference to the Present European Conflict." Dr. Henry S. Drinker, who is Chairman of the Institute committee on the Engineer's Reserve Corps movement, opened the evenings' talk with a short statement of the aims and advantages of the Students' summer military camps, fostered by the Society of the National Reserve Corps, he being a member of the Advisory Committee of University Presidents. The speakers of the evening were Col. William M. Black and Col. John Biddle, both of the Corps of Engineers, U. S. Army. Col. Black is the

Senior Colonel of the Army Engineer Corps and is at present Chief Engineer on the staff of Major Gen. Leonard M. Wood and is in charge of River and Harbor Improvements in New York Harbor. Col. Biddle is a high ranking Colonel of the Corps of Engineers, a graduate of the Army War College and a former member of the General Staff; more recently he has been attached to the Austrian Army as an observer, during the war now in progress. Other guests of the evening were Capt. E. D. Ardery, Capt. R. T. Coiner, Capt. T. M. Robins of the U. S. Army; Lieut. Col. E. W. Van C. Lucas, Majors Whitley and Conrow, and Captains Humphreys and Robinson, of the 22nd Corps of Engineers, N. Y. N. G. and Major Gen. O'Ryan, Commander of the State National Guard.

The duties of the engineer were pointed out in great detail by both Colonels Black and Biddle, and it was shown how dependent the modern army is on the Engineer Corps. The frontiers of belligerent European nations have been converted at present into practically continuous lines of trenches under the direction of engineer officers. The engineers charged with the maintenance and operation of lines of transport and communication, are as essential to the military operations as the men behind the guns. On the other hand, there are engineer troupes whose service is wholly destructive; bridges, railways, tunnels, roads, etc. must be demolished behind a retreating army. The difference between civilian and military engineering was pointed out, with especial reference to the value of the time element, and the weight to be given to methods and refinements which in military work expediency does not often justify.

The following list of books was recommended by Colonel Black for the civilian engineer, who desires to inform himself on matters pertaining to the subject discussed:

- Engineer Field Manual,¹
- Field Service Regulations,¹
- Infantry Drill Regulations,¹
- Military Railways,
- Technique of Modern Tactics,
- Art of Fortification,
- Annals of a Fortress,
- Fortification and Siegecraft (in Encyclopedia Britannica).

Some of these are included in the list issued by the War Department and published on pp. xiv to xvii of this *Bulletin*. A resolution was adopted, requesting that the Library of the Institute procure the books recommended by Colonel Black.

Colonel Lucas spoke of the work of his 22d Corps of Engineers and offered to designate a good company to be fathered and officered by the Institute, to specialize on the mining work required of the Engineers.

General O'Ryan spoke of the general need of coöperation of engineering societies with the National Guard movement and gave an interesting exposition of his successful efforts toward allaying the antipathy of labor unions to the Guard.

PERCY E. BARBOUR,
Chairman, Program Committee.

¹ Government Publications.

ST. LOUIS LOCAL SECTION

*Executive Committee*ARTHUR THACHER, *Chairman*R. A. BULL, *Vice-Chairman*W. E. McCOURT, *Secretary-Treasurer*, Washington University,
St. Louis, Mo.

H. A. BUEHLER

R. R. S. PARSONS

H. A. WHEELER

The St. Louis Section held a meeting in the Joplin District, Friday and Saturday, Oct. 29 and 30, 1915. The morning of Friday was spent in a general view of the district, with visits to the various properties and mills, among them the Bertha A slime plant and the Oronogo Circle property. After luncheon at Webb City, given by the Webb City operators, the Section visited the mines and mills of the American Zinc, Lead & Smelting Co. The dinner in the evening at Joplin, attended by 83 members and guests, was presided over by Arthur Thacher. The program follows:

For the Operators, C. T. ORR.

Geology of the District, H. A. BUEHLER.

Mining and Milling, C. A. WRIGHT.

The District Energy, GEORGE HAYLER.

Leases and Other Things, ALLEN McREYNOLDS.

The Institute, PHILIP N. MOORE.

On Saturday, as the guest of the coal operators in the Pittsburgh field, the Section visited the strip pit mining operations in that field.

The membership in the Section now numbers 154.

W. E. McCOURT, *Secretary*.

PENNSYLVANIA ANTHRACITE SECTION

*Executive Committee*R. V. NORRIS, *Chairman*CHARLES F. HUBER, *Vice-Chairman* EDWIN LUDLOW, *Vice-Chairman*N. J. RICHARDS, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*,CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines,
Wilkes-Barre, Pa.

DOUGLAS BUNTING

FRANK A. HILL

ALBERT B. JESSUP

RUFUS J. FOSTER

JOHN M. HUMPHREY

ROBERT A. QUINN

The Fall meeting of the Section was held at the Club Rooms of the Engineers' Society of Northeastern Pennsylvania, 415 Washington Ave., Scranton, Pa., at 8:45 p.m., Dec. 4, 1915.

An informal dinner, attended by 40 members and guests, preceded the meeting.

Chairman Norris presided at the meeting. Several proposals concerning the sources of increase of revenue for the American Institute of Mining Engineers were made. After some discussion, the members adopted a resolution recommending the increase of dues to \$15 per year and the members to receive the bound volumes of *Transactions*.

H. M. Crankshaw, member, then presented his paper, Modern Method of Mining and Ventilating Thick Pitching Seams. His paper was extremely interesting and elicited considerable discussion.

William Griffith, member, gave a brief informal talk on results of a number of experiments he has been conducting on small models of mine-roof supports. Mr. Griffith's talk was listened to with great interest. He was asked a number of questions concerning the relative supporting value of various types of supports discussed.

Hugh Archbald, member, presented a chart showing relative inside and outside delays in coal hoisting.

CHARLES ENZIAN, *Secretary*.

CHICAGO LOCAL SECTION

Executive Committee

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J. A. EDE, *Vice-Chairman*

H. W. NICHOLS, *Secretary*, 5213 Blackstone Ave., Chicago

F. K. COPELAND

G. M. DAVIDSON

The Fall Meeting of the Chicago Section has been postponed owing to the absence of the Secretary. The next meeting will be held Jan. 14, 1916, at the Chicago Engineers' Club, when there will be a talk by Dr. J. A. Mathews on Iron in Antiquity and Today.

AFFILIATED STUDENT SOCIETY NOTES

The School of Mines, of the University of Minnesota, held a meeting on Nov. 10, 1915, at which K. P. Swensen, formerly Professor in the Imperial Polytechnic Institute of Nanking, China, gave an illustrated lecture on mining and educational experiences in the Far East. Mr. Swensen spoke of the mining resources and opportunities of China, Japan, Korea and Manchuria, and related some of his experiences in those countries.

At the meeting held on Dec. 2, 1915, H. R. Bishop, of Cobalt, Ontario, gave a talk on Prospecting for Gold in Nicaragua.

ADOLPH DOVRE, *President*.

The Chemical Society of Tufts College, Medford, Mass., held a meeting on Nov. 22, 1915, at which 25 members, 3 members of the faculty and 9 alumni members were present. After a short business session the following papers were read:

Electrolytic Refining of Copper, by C. B. Town, '16.

Radio-activity of Chemical Elements, by H. L. Jones, '17.

Both of these papers were received with interest and the members of the faculty seemed pleased with the methods adopted by the Society in having students prepare papers.

LEO T. MURPHY, *Secretary*.

The Mining and Metallurgical Club of the University of Toronto made application for recognition as an Affiliated Student Society of the Institute, and at the November meeting of the Board of Directors, upon recommendation of the Committee on Affiliated Student Societies, this application was granted. The Affiliated Student Societies now number 26.

The officers of the Club are as follows:

H. E. T. HAULTAIN, *Honorary President*,
 B. A. McCRODAN, *President*,
 GEORGE HANMER, *Vice-President*,
 H. D. WALLACE, *Secretary-Treasurer*,
 J. W. CRANE, *2nd year Representative*,
 E. C. FAIRBROTHER, *1st year Representative*.

The University of Washington Mining Society, of Seattle, Wash., secured from the U. S. Bureau of Mines several reels of motion picture films illustrating mining and smelting operations and showed them on Nov. 8, 10 and 15. The subjects of these films were as follows:

Iron Mining Operations, showing exploration, stripping, underground mining, etc.

The Use of Explosives in Quarrying, showing drilling, firing and blasting.

Mine to Moulder, showing every step in the process.

Mining Iron Ore, showing drilling, blasting, hauling, timbering, etc.

The pictures were arranged with a view to emphasizing the methods of accident prevention.

On Nov. 10, the attendance was about 200, including nearly 100 children from one of the grammar schools. On Nov. 15, nearly 100 persons were present, composed chiefly of students and members of the faculty.

HENRY G. BOLTON.

The Students' Auxiliary Society at the University of Kansas, Lawrence, Kansas, has elected the following officers:

HARRY E. CRUM, *President*.

LELAND E. FISKE, *Vice-President*.

LAWRENCE E. COLE, *Secretary-Treasurer*.

This year the Mining Society meets on alternate weeks with the Geology Club. Each society has its own officers and organization, and arranges separate programs. So far this year the following talks have been given:

Mining Millions. By PROFESSOR TERRILL.

Roosevelt Drainage Tunnel. By H. E. FAIRCHILD.

Mining Schools of Today. By C. O. ANDERSON.

Secondary Enrichment. By H. P. CODY.

Relation of Geology to Mining. By S. F. KELLEY.

Southeast Missouri. By L. E. COLE.

LAWRENCE E. COLE, *Secretary-Treasurer*.

The Pick and Shovel Club of the Case School of Applied Science has elected the following officers for the current year:

R. D. SCHMIDT, *President*,

C. K. McDONALD, *Vice-President*,

W. P. SYKES, *Treasurer*,

F. C. McNUTT, *Secretary*,

C. E. DAVID, *Cor. Secretary*,

T. R. WILLSON, *Senator*,

L. S. KUEHN, *Historian*,

C. E. MIZER, *Serg. at Arms*.

At one of the first meetings of the college year F. M. Rapp, a graduate of the Case School and a member of the Pick and Shovel Club, told of his experiences in prospecting for diamonds in Africa. At another meeting Dr. Van Horn delivered a lecture on his trip through the gold mining district of Nevada, followed by a talk by Mr. Edwards a member of the faculty, on his trip through Canada and Minnesota. One evening was devoted to illustrated talks on the various practice-term trips taken by members of the Club. CARL E. DAVID, *Corresponding Secretary*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

- ELECTROTHERMIC SMELTING OF IRON ORES IN SWEDEN. Canada. Department of Mines. Mines Branch No. 344. Ottawa, 1915.
- EXPLOSIVES; THEIR MANUFACTURE, PROPERTIES, TESTS AND HISTORY. By Arthur Marshall. London, 1915.
- FRACTIONAL PRECIPITATION OF SOME ORE-FORMING COMPOUNDS AT MODERATE TEMPERATURES. Bull. No. 609, U. S. Geological Survey. Washington, 1915.
- EXPLOSIBILITY OF ACETYLENE. Technical Paper 112, U. S. Bureau of Mines. Washington, 1915.
- FUSED SILICA, BIBLIOGRAPHY OF. Compiled by A. E. Marshall & W. W. Winship. New York, Thermal Syndicate, Ltd., 1915. (Gift of Publishers.)
- INFLAMMABILITY OF MIXTURES OF GASOLINE VAPOR AND AIR. Technical Paper 115. U. S. Bureau of Mines. Washington, 1915.
- MINES AND MINING, ABSTRACTS OF CURRENT DECISIONS ON. October, 1914 to April, 1915. Bull. 101, U. S. Bureau of Mines. Washington, 1915.

- PHYSICAL METALLURGY, INTRODUCTION TO THE STUDY OF.** By Walter Rosenhain. New York, 1915.
- POCKETBOOK FOR MINERS AND METALLURGISTS COMPRISING RULES, FORMULAS, TABLES AND NOTES FOR USE IN FIELD AND OFFICE WORK.** Ed. 3, London, 1914.
- REPORT OF THE SELBY SMELTER COMMISSION.** Bull. 98, U. S. Bureau of Mines. Washington, 1915.
- SAFETY IN STONE QUARRYING.** Technical Paper 111, U. S. Bureau of Mines. Washington, 1915.

Chemistry and Physics

- ANALYSIS OF NATURAL GAS AND ILLUMINATING GAS BY FRACTIONAL DISTILLATION AT LOW TEMPERATURES AND PRESSURES.**
- BIBLIOGRAPHY OF THE CHEMISTRY OF GAS MANUFACTURE.** Technical Paper 120, U. S. Bureau of Mines. Washington, 1915.
- DETERMINATION OF NITROGEN IN COAL.** Technical Paper 64, U. S. Bureau of Mines. Washington, 1915.
- HANDBOOK OF CHEMISTRY AND PHYSICS.** Cleveland, The Chemical Rubber Co., 1914. (Gift of Publishers.) Price \$2.
- [NOTE.—Contains tables of chemical and physical constants, measures and units, wire tables and mathematical tables. Compiled by W. R. Veazy and Charles D. Hodgman of the Case School of Applied Science. A useful little handbook. W. P. C.]

Geology and Mineral Production

- AMERICAN IRON AND STEEL INSTITUTE.** Annual Statistical Report, 1914. Philadelphia, 1915.
- BIBLIOGRAPHY OF NORTH AMERICAN GEOLOGY FOR 1914 WITH SUBJECT INDEX.** Bull. 617, U. S. Geological Survey. Washington, 1915.
- ALASKA. MINERAL RESOURCES OF ALASKA.** Report on progress of Investigations in 1914. Bull. 622, U. S. Geological Survey. Washington, 1915.
- CALIFORNIA. GEOLOGY AND OIL RESOURCES OF THE WEST BORDER OF THE SAN JOAQUIN VALLEY NORTH OF COALINGA, CALIFORNIA.** Bull. 603, U. S. Geological Survey. Washington, 1915.
- CALIFORNIA. MINERAL PRODUCTION OF CALIFORNIA FOR 1914.** Bull. No. 70, California State Mining Bureau. California, 1915.
- FLORIDA. PHOSPHATE DEPOSITS OF FLORIDA.** Bull. 604, U. S. Geological Survey. Washington, 1915.
- ILLINOIS. COAL MINING IN ILLINOIS.** Bull. No. 13, State Geological Survey. Urbana, 1915.
- NEW YORK. MINING AND QUARRY INDUSTRY OF NEW YORK STATE.** N. Y. State Museum, Bull. 173, Albany, 1915. (Gift of University of State of New York.)
- ONTARIO. ONTARIO BUREAU OF MINES.** List of publications including reports, maps and bulletins, revised to Aug. 1, 1915. Bull. No. 25, Toronto, 1915. (Gift of Ontario Bureau of Mines.)
- WISCONSIN. PEAT RESOURCES OF WISCONSIN.** Bull. No. XLV, Wisconsin Geological and Natural History Survey. Madison, 1915.
- AUSTRALIA. CANBELEGO, BUDGERY AND BUDGERGAR MINES.** Part II of the Cobar Copper and Gold-field. Mineral Resources, No. 18, New South Wales. Department of Mines. Sydney, 1915.
- BRAZIL. RESOURCES OF MINAS GERAES (BRAZIL).** By A. F. Calvert. London-N. Y., 1915.
- NEW ZEALAND. GEOLOGY AND MINERAL RESOURCES OF THE BULLER-MOKIHINUI SUBDIVISION, WESTPORT DIVISION.** Bull. No. 17, n. s., New Zealand Geological Survey Branch. Wellington, 1915.
- PHILIPPINE ISLAND.** Mineral Resources, 1914. Manila, 1915.

General

- CARGO HANDLING METHODS AND APPLIANCES.** By H. McL. Harding. n. p. 1915. (Gift of author.)
- ELEMENTS OF HIGHWAY ENGINEERING.** By Arthur H. Blanchard. New York, J. Wiley & Sons, 1915. (Gift of Publishers.) Price \$3 net.
- [NOTE.—Designed for one semester course in highway engineering. W. P. C.]
- INTERNAL COMBUSTION ENGINES.** By Robert L. Streeter, New York, 1915.

DEWEY DECIMAL CLASSIFICATION. Ed. 9, 1915. N. Y. Lake Placid Clud, 1915.
 EASTMAN KODAK COMPANY. Abridged Scientific Publications from the Research
 Laboratory, 1913-14. Rochester, n. d. (Gift of Eastman Kodak Company.)

Gift of Hill Publishing Company

AMERICAN ZINC, LEAD AND SMELTING COMPANY. Annual Report, 1914.
 CHILE COPPER COMPANY. Descriptive circular.
 EXPLORAÇÃO DO RIO GRANDE E DE SEUS AFLUENTES, 1913. São Paulo, 1913.
 INTERNATIONAL NICKEL COMPANY. Annual Report, 1914.
 MOUNT MORGAN GOLD MINING COMPANY, LTD. Reports and statements of accounts,
 Nov. 29, 1914.
 NEW SOUTH WALES. Royal Commission. Report of on Mining Industry at Broken
 Hill, 1914. Sydney, 1914.
 SINTICH, VICTOR ESTUDIO DE LA FUERZA HIDRÁULICA DE LOS RIOS COLORADO, BLANCO
 Y RIO DE CHIUTA CON ANTEPROYECTOS PARA LA INSTALACIÓN DE CENTRALES
 HIDRO ELÉCTRICAS. Valparaíso, 1915.
 UNION OF SOUTH AFRICA. Department of Mines and Industries. Statistics for the
 month of August, 1915.
 SVERIGES OFFICIELLA STATISTIK. Bergshantering berättelse för ar 1914 av Kom-
 merskollegium.
 WALLAROO AND MOONTA MINING AND SMELTING COMPANY, LTD. Reports and state-
 ments of accounts. 25th Annual Report, 1914.
 WESTERN AUSTRALIA. Government Statistician's Office. Statistical Register, 1913.
 Perth, 1914.

Trade Catalogues

BUFF AND BUFF MANUFACTURING Co., Boston, Mass. Quality talk.
 CONNERSVILLE BLOWER Co., Connerville, Ind. Bull. 23, Connerville Boston type
 blowers. July, 1915.
 DEANE STEAM PUMP Co., Holyoke, Mass.
 Horizontal single acting triplex portable mine pumps. Pot valve pattern.
 Mine station pumps triplex single acting type—Pot valve pattern.
 Vertical triplex pot valve outside packed plunger mine station pumps.
 DENVER FIRE CLAY Co., Denver, Colo. Bull. 100, Metallurgical Clay goods.
 GREAT SOUTHERN LUMBER Co., Bogalusa, La. Select long leaf pine structural
 timbers.
 HARRISON SAFETY BOILER WORKS. Philadelphia, Pa. Finding and stopping waste
 in modern boiler rooms by the use of Cochrane meters. 1915. 68 pp.
 INGERSOLL-RAND Co., New York, N. Y.
 Form 9201. Calyx Core Drills. July, 1915.
 Catalog 76. Water lighted by compressed air. July, 1915.
 MASHEK ENGINEERING Co., New York, N. Y. Catalog. No. 3, Coal briquetting
 machinery and plants.
 SMITH ENGINEERING WORKS. Milwaukee, Wis. Sectional view of Telsmith Primary
 breaker.
 SPARTA IRON WORKS Co., Sparta, Wis. Sludge bucket. Nov., 1915.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Nov. 10 to Dec. 10, 1915:

BARTH, ERNEST, Petroleum Geol.....	Box 552, Tulsa, Okla.
BEROLZHEIMER, D. D., Chemist; Librarian, Barrett Mfg. Co.,	17 Battery Pl., New York, N. Y.
BLIEK, PIERRE FRANCISCO, Mgr., Compania Minera de Oruro,	Oruro, Bolivia, So. Amer.
BLOCK, JAMES A., Met.....	1076 1st Ave., Salt Lake City, Utah.
BULKLEY, RALPH G., Mine & Mill Supt., The Montezuma Mines & Mills Co.,	Montezuma, Colo.
BURRAGE, ALBERT CAMERON, JR., Met. Engr.; Director, 85 Ames Bldg., Boston, Mass.	
CABELL, CHARLES ARNOLD, Vice-Pres.....	Carbon Coal Co., Carbon, W. Va.
CHAPMAN, LEWIS CARL, Economic Geol.....	Producers Oil Co., Houston, Tex.
CHOFFEL, FLAVIEN AURELIEN, Mgr.....	Ingersoll-Rand Co., Paris, France.
COY, HARLEY A., Mine Supt.....	American Zinc Co., Mascot, Tenn.
DIETMANN, C. O. ADOLF, Cons. Engr., Beer, Sondheimer & Co., Inc.,	Santiago, Chile, So. Amer.
DOWELL, GRANT H., Asst. Genl. Mgr., Copper Queen Cons. Min. Co., Douglas, Ariz.	
DOYLE, DONALD B., Min. Engr., Care Mr. S. H. Ball, 71 Broadway, New York, N. Y.	
GARTUNG, FREDERICK HERMAN, Min. Engr.; Supt., Miami Zinc and Lead Co.,	Miami, Okla.
GLOBE, ALEXANDER R., Asst. Genl. Mgr., Canadian Mining & Finance Co.,	Timmins, Ont., Canada.
HEALY, RALPH LUCIUS, Min. Engr., Perseverance Mine,	Alaska Gastineau Gold Min. Co., Thane, Alaska.
LANTZ, GROVER B., Mining.....	Colton, Cal.
LAWRY, RAYMOND GEORGE, Engr. Colliery Construction, 7252 Yale Ave., Chicago, Ill.	
LOWDEN, HUGH B., Met.....	Secy., Colorado Iron Works Co., Denver, Colo.
MCCOY, JAMES COMLEY, Banker, Treas.....	Colorado Mining Co., Nayatt, R. I.
MCLAUGHLIN, WARNER, Met. Engr., Nipissing Mining Co., Cobalt, Ont., Canada.	
MAYCUMBER, WILLIAM R., Mill Supt., Care Tigre Mining Co., Esqueda, Sonora, Mex.	
MEAD, WARREN J., Prof. of Geology.....	University of Wisconsin, Madison, Wis.
SKEELS, FRANK HENRY, Min. Engr.; Mgr., Red Mountain Mine, Cornucopia, Ore.	
TEDDY, FREDERICK THOMAS, Min. Engr., Tennessee Copper Co.,	Ducktown, Polk Co., Tenn.
TEMPLEMAN, JOHN L., Lawyer, Mine Owner and Operator,	226 South Excelsior Ave., Butte, Mont.
URQUHART, CHARLES HOWARD, Met. Engr.....	Yates Hotel, Joplin, Mo.
ZEIGLER, WILLIAM LE VERNE, Mill Supt.....	Success Mining Co., Sunset, Idaho.

Associate Members

CANTWELL, HARRY JAMES, Lawyer....	Buckingham-Annex Hotel, St. Louis, Mo.
SCHIEREN, CHARLES ALBERT, Mfr., Pres., Charles A. Schieren Co.,	34 Ferry St., New York, N. Y.

Junior Members

LEVIS, ALFRED C.....	Portland Lodge, Victor, Colo.
NOTT, THOMAS EDWIN.....	4701 Ellsworth Ave., Pittsburgh, Pa.

Total Membership, Dec. 10, 1915.....5,318

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Nov. 10 to Dec. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

William R. Allen, Butte Mont.

Proposed by Walter Harvey Weed, Arthur B. Clark, J. D. Irving.

Born 1871, French Gulch, Mont. Montana Common Schools and College. 1902-06, Supt., French Gulch Dredging Co. 1903 to date, Pres., Allen Gold Min. Co. 1907 to date, Pres., French Gulch Min. Co. 1912 to date, Pres., Boston Montana Dev. Co. Engaged in mining for 25 years.

Present position: As above.

Rupert Hiram Bailey, Pierce, Ida.

Proposed by James McD. Porter, Robert N. Bell, L. K. Armstrong.

Born 1889, Nebraska City, Nebr. 1910, Grad., Univ. of Nebraska, B. S. 1911-13, General contracting and Bridge work, Genl. Mgr., Wilson-Bailey Construction Co., Walla Walla, Wash. 1913-15, Owner and operator, Pierce Co., Ltd., dredge at Pierce, Idaho. Mgr., Idaho Placer Mines Co., hydraulic placer operation, Newsome, Idaho.

Present position: Pres. & Genl. Mgr., Pierce Co., Ltd.

Frank Lincoln Bitler, Philadelphia, Pa.

Proposed by Henry S. Drinker, C. S. Robinson, John L. W. Birkinbine.

Born 1865, Philadelphia, Pa. Private, Public and Central High School, Philadelphia, B. A. in 1883 and M. A. in 1888. 1885-1915, Asst. to John Birkinbine, Cons. Engr. 1889-91, Expert Special Agent on Iron Ores for Eleventh Census. 1903-04, for Twelfth Census. 1906, Special Agent, U. S. Forest Service on Mine Timbers. 1889-1906, Engaged in collection of statistics of annual production of iron ores in U. S., also of manganese ores from 1896-1906. In these capacities visited iron ore mines in the principal iron ore producing States of the Union, also in Cuba. Collected and compiled figures in regard to the production of iron and manganese ores in this and foreign countries, also of iron ore reserves. Aided in iron ore mine arbitration, valuation and litigation, the extension of large iron and steel works, also water supplies.

Present position: Member, Birkinbine Engineering Offices.

Edward Mead Bordwell, Fairview, Nev.

Proposed by E. E. Carpenter, H. M. Alley, Robert G. Davies.

Born 1886, Jackson, Mich. 1893-1901, Grammar School. 1901-06, High School, Riverside, Cal. 1907-09; 1910-12, Univ. of Cal., Berkeley, Cal. 1904, 1905, 3 months each Greene Cananea Copper Co. 1906-07, Greene Cananea Copper Co. 1909-10, Forest Service, U. S. Government.

Present position: 1912 to date; Shiftboss in Mill and Cyanide plant, Nevada Hills Min. Co.

George O. Carpenter, St. Louis, Mo.

Proposed by Philip N. Moore, H. A. Wheeler, Elias G. Gatch.

Born 1852, Wakefield, Mass. 1863-65, Park Latin School. 1866-68, English High School. 1869-70, Mass. Inst. of Tech. 1871-77, various office positions, Chemist and Asst. Supt. 1877-90, Secy., Vice-Pres. and Pres., St. Louis Lead & Oil Co. 1891 to date, Director, Mgr., Vice-Pres., National Lead Co., St. Louis Branch. 1904 to date, Vice-Pres., St. Louis Smelt. & Ref. Co.

Present position: Vice-Pres., National Lead Co.; St. Louis Smelt. & Ref. Co.

André Chastel, Paris, France.

Proposed by F. Ledoux, A. R. Ledoux, E. Gybbon Spilsbury.

Born 1867, St. Etienne, France. 1888, Grad., School of Mines, St. Etienne, Civ. Engr. 1890-96, Eng., Ste. Minière et Metallurgique de Penarroya. 1898-99, Engr. of works, Cie des Mines d'Anzier. 1898-1908, Chief Engr., Ste. Minière et

Metallurgique de Penarroya; 1908-13, Director, Spanish Branch. 1913, Director Genl. Societe Miniere et Metallurgique de Penarroya, Paris, France.

Present position: Same as above.

Eric Kenneth Craig, Skidoo, Cal.

Proposed by E. A. Hersam, W. S. Morley, Roy Leach, J. H. Cooper.

Born 1889, Piedmont, Cal. 1904-08, Oakland High School, Oakland, Cal. 1909-14, Univ. of Cal., B. S. 1911, Mucker and cyanide-man, North Star Mines Co., Grass Valley, Cal. 1914-15, Sampler, timberman, miner, Inspiration Cons. Copper Co.

Present position: Head Amalgamator, Skidoo Mines Co.

P. H. Dever, Nanticoke, Pa.

Proposed by H. G. Davis, R. V. Norris, Charles Enzian.

Born 1890, Phoenixville, Pa., Public Schools. 1891, Wyoming Seminary. I. C. School of Mines. 1882, Franklin Coal Co. 1883-84, Lehigh and Wilkes-Barre Coal Co. 1885-88, D. & H. Coal Co. Also a short time for Kingston Coal Co.

Present position: 1887 to date; Asst. Dist. Supt., Delaware, Lackawanna and Western R. R. Co.

Arthur Wiley Evans, Petros, Tenn.

Proposed by A. H. Wood, Edward H. Coxe, F. H. Clymer.

Born 1870, Rockwood, Tenn. 1881-82, High School, Dalton, Ga. 1882-86, Dade Normal School, Trenton, Ga. 1890-92, Univ. of Ohio, Columbus, O. 1892-94, Asst. Min. Engr., Georgia Min., Mfg. and Investment Co. 1895-97, Min. Engr., Brushy Mt. Coal Mines. 1897-98, Supt., Durham Coal and Coke Co. 1899-1900, Supt., New Soddy Coal Co., Soddy, Tenn. 1901-03, Supt., Brushy Mt. Coal Mines. 1903-10, Mgr., Lockout Fuel Co. 1910-12, Dist. Mine Inspector, State of Tenn. 1912-14, Supt., Brushy Mt. Coal Mines. 1914-15, Chief Engr., Oneida and Western R. R.

Present position: Supt., Brushy Mt. Coal Co.

Cicero Coelho de Faria, Rio de Janeiro, Brazil, S. A.

Proposed by Horace E. Williams, Euzebio Paulo de Oliveira, W. L. Saunders.

Born 1884, Bahia, Brazil. 1894-1903, Grad., College and High Schools, Bahia. 1905, Higher course, Polytechnic School, Bahia, where had good marks and passed through an incomplete course. 1910, Engineer-Geographer. 1912, Grad., State Engr., special course, Engineering School, Rio de Janeiro. 1906-12, has always been working and studying. 1907, Timbo Railway; (Survey) Railway, Oeste de Minas y Companhia E. F. Victoria a Diamantina. 1908, Chief of Survey, Leopoldina and Northwest of Brazil Railroad Co. 1910, Leveling and Surveying, International Transparaguayan Ry., and Carrocavel-Petropolis Ry. (Rio de Janeiro). 1912, Fiscal engineer Federal Railways.

Present position: Fiscal engineer of railways, attached to the office of the Minister of Railroads of Brazil.

Francis J. Gibbons, Shultz, Ariz.

Proposed by W. G. McBride, C. Legrand, G. H. Dowell.

Born 1882, West Chester, Pa. 1889-1900, Grade & High Schools, Kirksville, Mo. 1900-02, State Normal School, Kirksville, Mo. 1902-05, Teaching in High School. 1905-06, Missouri School of Mines, Rolla, Mo. 1907 (summer) Aymour Institute, Chicago. 1906-07, Engrg. Offices, Cananea Cons. Copper Co. and Cananea Central Copper Co., Cananea, Son., Mex. 1907-08, City Engineer's Office, Bisbee, Ariz. 1908-11, Chief Engr., Great Western Copper Co., Courtland, Ariz.

Present position: 1911 to date; Asst. Supt., Young Bros.

John Tinker Glidden, Cerro de Pasco, Peru, S. A.

Proposed by Paul S. Couldrey, Charles E. Locke, Edward E. Bugbee.

Born 1883, Cambridge, Mass. 1905, Mass. Inst. of Technology, S. B. General Mining work since graduation.

Present position: Mine Supt., Cerro de Pasco Mining Co.

Frank Howard Franklin Hampton, Rancagua, Chile, So. Amer.

Proposed by H. A. Guess, R. K. Stockwell, H. C. Bellinger.

Born 1884, Ledbury, England. 1888-93, Abbey House School, Ledbury, England. 1893-99, Grammar School, Ledbury, England. 1900-04, Gloucester Municipal School of Engineering, 2nd and 1st Class, Board of Education Certificates. 1900-06, Pupil of Messrs. Summers and Scott, Ltd., Gloucester, England. 1906-07, In shops of Messrs. Vickers, Ltd., Erith, England. 1907-08, Draftsman, New Conveyor Co., Birmingham, England. 1908-09, Draftsman, Heenan & Froude, Ltd., Worcester, England. 1909-13, Draftsman, Fraser & Chalmers, Ltd., Erith, England. 1913-14, Draftsman, W. G. Perkins, Cons. Engr., London, England.

Present position: 1914 to date; Asst. Chief Draftsman, Braden Copper Co.

R. A. Harder, Aurora, Nev.

Proposed by E. A. Julian, Charles C. Starr, James O. Greenan.

Born 1886, Kimball, So. Dak. 1910, Mackay School of Mines, Univ. of Nev. 1910, General field work, examinations, etc. 1911, Geol. work, Nevada Hills Min. Co. 1912-13, Underground Supt., Nevada Hills Min. Co. 1914, Genl. Supt., Nevada Hills Min. Co.

Present position: 1914 to date; Genl. Supt., Aurora Cons. Mines Co.

Harry Arthur Harper, Rancagua, Chile, So. Amer.

Proposed by H. A. Guess, R. K. Stockwell, Lester E. Grant.

Born 1888, Ironwood, Mich. General High School, Calumet. 1907, Michigan College of Mines, Houghton, Mich. 1910, Chem., Castil Min. Co., Ramsay, Mich. 1910, Cons. Engr., Iron River Dist., Mich. 1911, Chief Engr., Gogebec County Road Commission, Bessemer, Mich. 1912, Contract on road work, Bessemer, Mich.

Present position: 1913 to date; Foreman, Tailings Retreatment Flotation Concentrator, Braden Copper Co.

Mariano R. Hartmann, La Granga, Chile, So. Amer.

Proposed by Eric V. Burnett, C. S. Witherell, Henry Hay.

Born 1878, La Tirana, Tarapaca, Chile. 1888-94, Instituto Internacional, Santiago de Chile. 1895-1908, in charge of a patent lixiviation plant and installation of same in several silver mines, Bolivia. 1911-13, Mgr. and Engr., Sindicato Minera "El Tobar" de Collahuasi, Iquique, Chile.

Present position: Mgr. La Nueva Compania de Cobre de Copaquiere and Owner and Mgr., Challacollo Silver Mines and Establishment.

George Edward Hayes, Mace, Idaho.

Proposed by Rush J. White, James F. McCarthy, Ernest G. Gnaedinger, L. K. Armstrong.

Born 1875, Blarney (Cork). 1894, Camborne School Mines, Cornwall, England. Technical Mining and Metallurgy. 1890, Broken Hill Silver Mines, N. S. W., Aust. 1893, Coolgardie W. A. Hayes; Find and Hit or Miss. 1898, Mine Captain, Rezendi Mines, Umtali, So. Africa. 1907, Head Surveyor, Great Indian Peninsular Ry., Bombay, India. 1910, Burma Mines Ry. Smelt. Co. 1912, Cons. Engr., Rayfield Gold & Tin Areas, N. Nigeria. 1913-15, Cons. Engr. and Geol., 105 8th Ave., W. Calgary, Alta., Canada.

Present position: Hecla Mining Co., Burke, Idaho.

James William Hutchinson, Goldfield, Nev.

Proposed by E. A. Julian, Charles C. Starr, James O. Greenan.

Born 1881, Oxford, Miss. 1898-1901, Univ. of Mississippi. 1904-05, Laborer, various mines in Colorado. 1906, Laborer, various mills in Colorado. 1906-07, Tonopah Min. Co., Tonopah Nev. 1907, Tonopah Belmont Dev. Co., Tonopah, Nev. 1908-13, Mill Supt. 1913-14, Asst. Genl. Mgr., Goldfield Cons. Mines Co., Goldfield, Nev.

Present position: Genl. Mgr., Goldfield Cons. Mines Co.

Frederick Joseph Joubert, Camptonville, Yuba Co., Cal.

Proposed by Spencer C. Browne, R. H. Elliott, George O. Scarfe.

Born 1882, Camptonville. 1906, Grad., Univ. of Cal. 1909-14, Supt., Snowden Hill Drift Gravel Mine, owned by American Min. & Dev. Co., Boston. Also surveying for neighboring mines.

Present position: Supt., Depot Hill Mine.

J. H. Jowett, New York, N. Y.

Proposed by W. L. Saunders, Henry Lang, George A. Howells.

Born 1874, Cleveland, O. 1891, Grad., Cleveland High School and Manual Training School, Cleveland, O. 1891-93, Learning Machinist Trade, Cleveland-Shipbuilding Co. 1894-96, Drafting, Steel Motor Co., Johnstown, Pa., and McMyler Mfg. Co., Cleveland, O. 1897, Draftsman, Ingersoll Sergeant Drill Co. 1900-05, Salesman, Ingersoll Sergeant Drill Co.

Present position: 1905 to date; Genl. Sales Mgr., Ingersoll Rand Co. and A. S. Cameron Steam Pump Works.

George W. Kays, Spokane, Wash.

Proposed by L. K. Armstrong, Charles H. Goodsell, D. C. Livingston.

Born 1877, Abingdon, Ill. 1901, Grad., Idaho School of Mines, Moscow, Ida. 1901-03, Assayer, Morning Mine, Mullan, Ida. 1903-06, Smelter Representative, Federal Min. & Smelt. Co., Wallace, Ida. 1906-08, Mgr., Hosey Mine, Patagonia, Ariz. 1908-10, Chief Engr., Rawhide Coalition, Rawhide, Nev. 1910-11, Mill Supt., Guanajuato Dev. Co., Guanajuato, Mex. 1911-12, Examination of mines in

Mexico. 1912-13, Mill Foreman, San Poil Co., Republic, Wash. 1913-14, Constr. Engr., Canadian Consolidated, Rossland, B. C.

Present position: Operating Lease on Surprise Mine.

John Byrns Lain, Goldfield, Nev.

Proposed by K. M. Simpson, Willis Lawrence, J. K. Turner.

Born 1885, Mansfield, Tex. 1902, Grad., High School, Mineral Wells, Tex. 1904, Asst. Assayer, Humboldt Smelter, Humboldt, Ariz. 1905-06, Millman, Montezuma Copper Co., Nacozari, Mexico, and Cananea Consolidated, Cananea, Mex. 1907, Millman, cyanide plant, Desert Power & Milling Co., Tonopah, Nev. 1908, Millman, cyanide plant, Montgomery-Shoshone Mill, Rhyolite, Nev.

Present position: 1908 to date; Mill Supt., Goldfield Cons. Mill. & Transportation Co.

George Robert Lehman, Miami, Ariz.

Proposed by John Langton, C. E. Mills, C. E. Arnold.

Born 1877, Frankford, Ind. 1892-96, Santa Barbara High School. 1896-1901, Univ. of Cal., B. S. 1899, Practical underground experience, Mother Lode and Nevada Co., Cal. 1902, Asst. Engr., North Star Mines Co., Grass Valley, Cal. 1903-04, Asst. Supt., Smelter, Fernando Min. Co., San Fernando, Dgo., Mex. 1905-06, Mine Engr., New York & Honduras Rosario Min. Co., San Juancito, Honduras, C. A. 1907-12, Chief Engr., Warrior Copper Co., Warrior Development Co., Live Oak Development Co., Cordova Copper Co., Globe Miami Min. District, Ariz.

Present position: 1912 to date; Chief Engr., Inspiration Cons. Copper Co.

Alexander William MacNichol, San Francisco, Cal.

Proposed by Charles W. McMeekin, Charles C. Derby, Thomas V. Reeves.

Born 1885, San Francisco, Cal. 1908, Univ. of Cal., B. S. 1908-09, Miner, Goldfield Cons. 1909, Miner, North Star. 1910, Bookkeeping, Surveying, Mammoth Copper Mine Quartz Hill. 1911, Lightner Gold Min. Co., Nevada Hills. 1912, Assaying, Bookkeeping, Surveying, Croesus Gold Min. & Mill. Co. 1913, Supt., Rainbow Mine. 1914, Supt., Gold Canon Mine.

Present position: Foreman, Copper King Mine.

Charles F. O. Merriam, Wallace, Idaho.

Proposed by J. McD. Porter, Rush J. White, Robert N. Bell.

Born 1875, New Brunswick. Two years' college course in the Univ. of Minnesota and of Idaho. Mining Engineer for 15 years, Wallace, Idaho. 1906; Mgr. Stewart Mine, and since Consulting Engineer for this company. Con. Engr., Success Mine since 1905.

Present position: Consulting Engineer for a great many prospects.

Shin Nakagawa, Tokyo, Japan.

Proposed by Edward L. Young, M. Otagawa, S. Yamanouchi.

Born 1885, Tokyo, Japan. 1905-07, First Higher School, Tokyo. 1907-10, Dept. of Min. and Met., Tokyo Imperial Univ. 1910-12, Asst. Inspector of mines.

Present position: 1912 to date; Min. Engr., Imperial Bureau of Mines.

Percy Melrose Newhall, Johannesburg, So. Africa.

Proposed by D. Wilkinson, C. E. Knecht, R. V. Warriner

Born 1873, Kirby, Vt. To 1892, Public Schools, Vermont & California. 1892-94, Teaching in Public Schools. 1894-98, Univ. of Cal., B. S. 1901-02, Univ. of Cal., Geology and Mining. 1898-99, U. S. Army, Vol. Engrs. 1899-1901, U. S. Coast & Geodetic Survey (Alaska) U. S. Navy, Hydrographic Dept. (Panama Mexican Coast & Guam). 1902-05, Sampler, Surveyor, Const. Engr. & Genl. Asst. to Mgr., Witwatersrand Deep, Ltd. & Knight Central, Ltd.

Present position: 1905 to date; Asst. Cons. Engr., S. Neumann & Co.

Louis S. Panyity, Columbus, O.

Proposed by Roswell H. Johnson, L. G. Huntley, H. B. Meller.

Born 1890, Budapest, Hungary. 1900-02, Reformed Gymnasium of Budapest. 1905-09, N. Y. High School of Commerce. 1911-14, Univ. of Pittsburgh, School of Mines. 1913, Field Instructor in Surveying, Univ. of Pittsburgh. 1914, Fayette Co. Gas Co., Pa.; United Fuel Gas Co., Charleston, W. Va.

Present position: Geol., Ohio Fuel Supply Co.

Frederick H. Penn, Tonopah, Nev.

Proposed by A. H. Jones, W. H. Blackburn, Frederick Bradshaw.

Born 1884, St. Louis, Mo. 1889-1901, Grade and High Schools, St. Louis and the Cripple Creek District; special studies during summers of 1900-01 and up to 1905. 1900-01 (summer), Assayers Asst. 1901-05, Min., Mine Surveying, Mill. and Assay. C. C. Dist. and vicinity. 1905-07, Asst. Supt., development work,

Wolframite Gold M. & M. Co., Dale, Colo. 1907-08, Contract Min. Wonder, Nev., and millman, Belmont Mill. Co. and Desert Power and Mill Co., Millers, Nev. 1908-09, Erecting and starting 150 ton Tailing plant, Sacramento Mine, Mercur, Utah. 1909-10, Mill, Cherry Creek, Lane City and Goldfield, Nev.

Present position: 1910 to date; Mill Foreman, Belmont Mill. Co.

Irving C. Purlington, Wallace, Ida.

Proposed by J. McD. Porter, Rush J. White, Robert N. Bell.

Born 1882, Minnesota. 1899, Grad., High School. 1900-03, Rodman and levelman, Spokane Valley Land & Water Co. 1904, Levelman, Washington Water Power Co. 1905-08, Transitman, Arthur A. Booth, E. M. & U. S. Mine. 1909-10, Federal Min. & Smelt Co., Wallace, Idaho. 1911-15, General min. and survey. practice, Coeur d'Alene Min. Dist.

Present position: Independent engineering practice.

Edward L. Queen, White Oaks, N. M.

Proposed by Charles A. Mitke, Arthur Notman, LeRoy B. Mitchell.

Born 1881, Pueblo, Colo. 1887-91, Common Schools, San Antonio, N. M. 1891-97, Common Schools, White Oaks, N. M. 1897-1915, Practical Mining. 1897-1903, Old Abe Co., White Oaks, N. M. 1903-04, Prospecting for mineral. 1905-15, Part owner and Supt. of Mines, Wild Cat Leasing Co., White Oaks, N. M.

Present position: Interested in and operating mining property.

Harold J. Rahilly, Pittsburgh, Pa.

Proposed by Van H. Manning, Albert H. Fay, J. D. Davis.

Born 1889, Lake City, Minn. 1896-1903, Madison Grammar School; 1903-07, Central High School, Minneapolis, Minn. 1907-11, School of Mines, Univ. of Minn., E. M. 1911-12, Min. Engr., Oliver Iron Min. Co., Duluth, Minn. 1912-13, Min. Engr., Repath & McGregor, Douglas, Ariz. 1913, Constr. Engr., Calumet & Arizona Min. Co., Douglas, Ariz. 1913, Const. Engr., Commonwealth Min. & Mill. Co., Pearce, Ariz. 1913-14, Cons. Engr., Douglas, Ariz. 1914-15, Supt., Minneapolis Copper Co., Cumpas, Son., Mex. 1915, Min. Engr., U. S. General Land Dept., Kink, Alaska.

Present position: Junior Min. Engr., Bureau of Mines.

Homer Irving Reynolds, Wallace, Ida.

Proposed by Rush J. White, Fred W. Callaway, L. K. Armstrong.

Born 1871, Spring Valley, Colo. 1891, Left Military Academy, Wash. 1887, Asst. Surveyor, minerals and lands, New Mex., Tex., Southern Colo. 1897, Course assaying, Butte, Mont. 1887, Began Geological study under O. Freeman in Colo. 1906, Cons. Engr., Takilma, Ore. 1907, Examining Engineer, Nev. 1909, Mgr., Scranton Mine, Alta.

Present position: 1900 to date; General practice in mine engineering and directing work. Examinations of mines, prospects and geological work in all districts of Idaho, some districts in all Pacific Coast States, Alaska, B. C., Alberta, Rocky Mountain States and Alabama.

Jack B. Riefkin, New York, N. Y.

Proposed by Van H. Manning, Frank A. Ray, Albert H. Fay, Oliver Bowles.

Born 1888, Keir, Russia. 1905, Public School, Newport. 1906-09, Kentucky State Univ. 1909-12, Ohio State Univ., E. M. 1912-13, Two years' experience, American Coal Co., McComas, W. Va.

Present position: 1913 to date; Engr., U. S. Bureau of Mines.

Edward J. Roberts, Twin Falls, Idaho.

Proposed by Walter Fitch, Howard Fitch, G. W. Crane.

Born 1882, Eureka, Nev. 1900-03, Univ. of Nevada. 1904, Survey and assay, Tonopah, Nev. 1905-06, Reporting on property, T. J. Lynch, T. E. Edwards and others, Nev. 1906-08, Operating for E. J. Roberts and associates. Supt., Nevada Alpine Co. 1909-14, Engrg. & reporting on properties, Office, Twin Falls, Idaho. 1915, With C. F. Sherwood, Butte, Mont. 1915, Engr., J. J. Daly Co., Salt Lake City, Utah.

Present position: Pres. & Genl. Mgr., Daily Cons. Mines Co.

W. Murray Sanders, Duquesne, Ariz.

Proposed by H. T. Herr, C. A. H. de Saulles, H. A. Prosser.

Born 1878, Roselle, N. J. 1899, Grad., Sheffield Scientific School, Yale Univ., Ph. B. One graduate year in Chemical Research. 1901-02, Chemist, Laboratory, General Chemical Co., Laurel Hill, L. I. 1903-04, Independent metallurgical investigations. 1904-05, Met., Lanyon Zinc Co., Iola, Kans. 1905-14, Genl. Mgr.,

Sanders Ore Operating Co., Marion, Ky. Flotation, concentration and general min. and mill.

Present position: Asst. Gen. Mgr., Duquesne Min. & Red. Co.

Peter Marseilles Saxman, Colver, Pa.

Proposed by Eduard V. D'Inwilliers, Walter Gilman, Edwin F. Saxman.

Born 1889, Latrobe, Pa. 1902-06, St. Lukes School, Wayne, Pa. 1906-09, Univ. of Penn., Philadelphia, Pa. 1909-10, Columbia Univ., New York, N. Y. 1906-09, Between Collegiate terms different minor positions with Saxman Coal and Coke Co., Saxman, W. Va. 1910-11, Blast Furnace Dept., The Central Iron and Steel Co., Harrisburg, Pa. 1911-15, Min. Eng. and Genl. Supt., The Saxman Coal and Coke Co., Saxman, W. Va.

Present position: Supt., The Ebensburg Coal Co.

Harry G. Schaupp, New York, N. Y.

Proposed by William Campbell, Robert Peele, W. A. Scheuch.

Born 1874, Dubuque, Iowa. Common School education. 1895-99, Min. and prospect., western U. S. 1899-10, Mining in Alaska. 1911, Min. examinations, So. Africa.

Present position: 1912 to date; Asst. Engr., Cie. des Chemins de Fer du Congo Superieur aux Grands Lacs Africains, Belgian Congo.

Walter A. Schmidt, Los Angeles, Cal.

Proposed by Ira B. Joralemon, John C. Greenway, L. D. Ricketts.

Born 1883, Los Angeles, Cal. 1906, Univ. of Cal., B. S. 1906-09, Laboratory and business experience. 1909-10, Petroleum—assisting development electrical dehydration process. 1910-11, Assisting development electrical precipitation process, Cottrell Process.

Present position: 1912 to date; Genl. Mgr., Western Precipitation Co.

Robert Antoine Schmucker, Rancagua, Chile, So. Amer.

Proposed by R. K. Stockwell, Lester E. Grant, John P. Chadwick.

Born 1885, Philadelphia, Pa. 1904-05; 1912-15, Mass. Inst. of Tech., B. Sc. 1905, Draftsman, Clinton Wire Cloth Co., Clinton, Mass. 1906-08, Draftsman & Machinist, Colorado Drill & Mach. Co., Denver, Colo. 1908-10, Met. Draftsman, J. E. & W. E. Greenawalt. 1908-10, Assayer & Chem., United Rico Mines Co., Rico, Colo. 1910-12, Draftsman, Steptoe Valley Smelt. & Min. Co., McGill, Nev.

Present position: Draftsman, Braden Copper Co.

Hubert N. Stronck, Chicago, Ill.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1892, Le Mars, Iowa. 1907-09, Armour Inst. of Tech. 1909-13, Colorado School of Mines, E. M. 1912, Ray Cons. Copper Co. 1913, Big Horn Collieries, Crosby, Wyo. 1913-14, Associated with James F. Boyle, New Haven, Conn. as Cons. Engr. of work in Germany; headquarters in Berlin. Lecturer on Min. and Met. Efficiency in Germany.

Present position: Senior Engr., Min. Dept., staff of L. V. Estes, Inc.

Albert Emmanuel Swartz, Rancagua, Chile, So. Amer.

Proposed by R. K. Stockwell, W. J. Turner, Lester E. Grant.

Born 1885, McKeesport, Pa. 1905, International Correspondence School, Mech. Draw. and Math. American School of Correspondence, Mech. and Elec. Engrg. 1901-06, Sheet steel mill worker, American Tin Plate Co., McKeesport, Pa. 1906-09, Draftsman, National Tube Co., McKeesport, Pa. 1909, Draftsman, United Engrg. and Fdry. Co., Pittsburgh, Pa. 1909-10, Draftsman, National Tube Co., Lorain, O. 1913-14, Draftsman, Youngstown Sheet & Tube Co., Youngstown, O.

Present position: Draftsman, Braden Copper Co.

Michael J. Sweeny, Spokane, Wash.

Proposed by D. C. Livingston, L. K. Armstrong, J. W. McBride.

Born 1866, Paterson, N. J. 1873-79, Public Schools; 1879-83, St. Ignatius College, San Francisco, Cal. 1883 to date; Practical School of experience. 1885-90, Placer mining, prospecting and developing placer and quartz, "North Side" of Coeur d'Alene Dist., Ida. 1890-92, All departments up to Foreman Lead Concentrator, Last Chance Mill, Wardner, Ida. 1893-98, Part owner and Mgr., Silver Bell Mine, McGuigan Basin, B. C. 1898-1902, Asst. Genl. Mgr., later Genl. Mgr., Buffalo Hump Syndicate and Big Buffalo Min. Co., Idaho Co., Ida. 1908-10, Associate Mgr., Cracker Jack Mine and Mill, Idaho Co., Ida.

Present position: Pres. & Genl. Mgr., Rex Cons. Min. Co.

J. R. Thill, Rancagua, Chile, So. Amer.

Proposed by H. A. Guess, R. K. Stockwell, W. J. Turner.

Born 1884, Lake Linden, Mich. 1899-1903, All Hallows College, Salt Lake City, Utah. 1903-08, Smelter Mechanic, Boston & Montana Smelter. 1908-09, Constr. U. S. Smelt. & Ref. Co. 1909-11, Smelt. constr. Int. Smelt. Co. 1911-12, Erect. Engr., Inspiration Cons. Copper Co. 1913-14, Charge of shops and mech. work, Smelter, Garfield Smelt. Co.

Present position: Asst. Master Mechanic, Braden Copper Co.

Jerrold Roscoe Underwood, Granby, Mo.

Proposed by Edwin T. Perkins, Nelson B. Gatch, H. A. Buehler.

Born 1878, Joplin, Mo. 1899, Missouri School of Mines, B. S. 1903, Missouri School of Mines, E. M. 1899-1903, Chem. Granby M. & S. Co., Granby, Mo. 1903-05, Ore buyer, Supt., Lead Smelter and ore dressing Dept., Granby. M. & S. Co. 1905-06, Asst. Supt. and Supt., Lead Smelter, St. Louis Smelt. & Ref. Co., Collinsville, Ill.

Present position: 1906 to date; Mine Operator and Owner, Granby, Mo.

Frank Sherman Washburn, New York, N. Y.

Proposed by W. L. Saunders, George A. Howells, George T. Cousins.

Born 1860, Centralia, Ill. 1878-1885, Engrg. Course, Cornell Univ., C. E. 1879-80; 1883-84; 1885-88, C. & N. W. Ry. Co. 1888-89, U. S. Y. & T. Co., Chicago, Ill. 1890-1900, Engineering and Contracting, developing into an average business of \$2,500,000 per annum. Organized various projects.

Present position: Pres., Goodman Mfg. Co. of Chicago.

Robert C. Weed, Duluth, Minn.

Proposed by F. C. Langenberg, Albert Sauveur, H. M. Boylston.

Born 1887, Starkville, N. Y. 1909, Ph. B. 1910, Browne Univ., M. S. 1913, Harvard Univ., M. E. 1913-14, Test. Dept., Anaconda Copper Min. Co. 1914-15, Met. laboratory, Illinois Steel Co., Gary, Ind. 1915, Chief Inspector, Gold Rolling-mill, Stanley Hardware Co., New Britain, Conn.

Present position: Met., Minnesota Steel Co.

Alfred Helm Westall, Fairview, Nev.

Proposed by E. A. Julian, Charles C. Starr, James C. Greenan.

Born 1883, Scales, Sierra Co., Cal. 1900-04, Palo Alto High School. 1904-08, Grad., Univ. of Nev., Min. Dept. 1908-09, Montan Tonopah Min. Co., Tonopah, Nev. 1910-11, Nevada Power and Red. Works, Dayton, Nev. 1911-15, Saline Valley Salt Co., Bishop, Cal.

Present position: Supt., Nevada Hills Min. Co.

Henry Arthur White, Springs, Transvaal, So. Africa.

Proposed by W. A. Caldecott, Walford R. Dowling, James E. Thomas.

Born 1872, Gloucester, England. Member, Institute of Mining and Metallurgy, Vice-Pres., Chem., Met., and Min. Soc. of S. Africa. Twenty years metallurgical experience on the Rand.

Present position: Cons. Met., A. Goenz. & Co., Ltd.

Homer L. Williams, Tonopah, Nev.

Proposed by John G. Kirchen, John L. Dynan, W. H. Blackburn.

Born 1887, Butte Co., Cal. 1909, Univ. of Nevada, B. S. 1909-11, Assayer, Montgomery Shoshone Min. Co., Rhyolite, Nev. 1911-14, Foreman, West End Mill, Tonopah, Nev. 1914, Mill Supt., Pioneer Cons. Min. Co., Pioneer, Nev.

Present position: 1914 to date; Min. Engr., Tonopah Ext. Min. Co.

Michael Thomas Williams, Westonia, W. Aust.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1871, Victoria, Aust. Matriculation Standard, Corporate High School, Bendigo, Vict. 1899, Certificate of Competency, Mine Mgr., Bendigo School of Mines, Vict., Aust. 1892-1903, Various positions. 1903-04, Acting Mine Mgr., Johnson's Reef G. M. Co., Bendigo, Vict. 1904-05, Mine Mgr., Watkins Tin Min. Co., N. S. W. 1905-10, Mine Mgr., Arthur's Creek G. M. Co., Vict. 1910-13, Mine Mgr., Central Cookman's G. M. Co., Maldon, Vict.

Present position: 1913 to date; Genl. Mgr., Edna May G. M. Co.

Motoyoshi Yamashita, Tajima, Japan.

Proposed by Edward L. Young, Lewis A. Jeffs, W. A. Wilson.

Born 1883, Sonobe, Japan. 1888-1896, Primary School. 1896-1901, Middle School. 1902-05, Higher Middle School. 1905-08, Kyoto Imperial Univ. Since graduation, Min. Engr., Mitsui-Bishi & Co.

Present position: Min. Engr., Ikuno Mine of Mitsu-Bishi & Co.

Associate Members

Axel Wilhelm Cajander, Rancagua, Chile, So. Amer.

Proposed by R. K. Stockwell, H. A. Guess, Lester E. Grant.

Born 1889, Uddevalla, Sweden. 1906-10, Chalmersska Institutet, Gothenburg, Sweden. 1910-12, Foreman, Munkedal's Paper Mill, Sweden. 1912-13, Mechanic, Karlstad Mfg. Co., Sweden.

Present position: Draftsman, Braden Copper Co.

Frank Carroll, Juneau, Alaska.

Proposed by R. W. Rusterholz, E. M. Norris, John R. Bartlett.

Born 1879, Hiawasse, Ga. Three years High School, Trinity, Colo. 1900-01, Deadwood Mines, Ophir, Colo. 1902, Atlas Min. Co., Ouray, Colo. 1904-05, San Pedro Min. Co., Ouray, Colo. 1906-07, Lena Min. Co., Fairview, Nev. 1907-08, El Porvenir M. Co., Parral, Mex.

Present position: 1908 to date; Mgr., Ingersoll Rand Co.

Stephen G. Murray, Seattle, Wash.

Proposed by R. W. Rusterholz, E. M. Norris, John R. Bartlett.

Born 1856, Chatham, N. J. Common School; five years, West Rutland, Vt. 1881, Steam Stone Cutter Co., Rutland, Vt. 1884, Ingersoll Rock Drill Co., Ingersoll Sergeant Drill Co., Ingersoll Rand Co. One year, Supt., Stone Quarry, J. P. Fault & Co.

Present position: Representative, Ingersoll Rand Co.

Richard P. Thomas, Kingston, Pa.

Proposed by H. G. Davis, R. V. Norris, Charles Enzian.

Born 1873, Swansea. Grad., Coal Mining Course, I. C. S., Scranton, Pa.

Present position: Asst. Dist. Supt., Delaware, Lackawanna and Western Co.

Frank Blair Turpin, San Francisco, Cal.

Proposed by Edward H. Benjamin, Stuart L. Rawlings, B. Stanley Revett.

Born 1866, Olympia. Have been interested in mining in Mexico and Nevada for 20 years.

Present position: Mine owner in Old Mexico.

Junior Members

Fan Chen, Golden, Colo.

Proposed by William B. Phillips, F. W. Traphagen, W. G. Haldane.

Born 1893, Hengchowfu, Hunan, China. 1904-08, Hunan School, Wuchang, China. 1908-11, School of Languages, Wuchang, China. 1911-13, Tsing-hua College, Peking, China.

Present position: 1913 to date; Senior Student, Colorado School of Mines.

Frank Edward Briber, Golden, Colo.

Proposed by William B. Phillips, Harry J. Wolf, F. W. Traphagen.

Born 1890, Johnstown, Pa. 1908-12, West Denver High School.

Present position: Student, Colorado School of Mines.

Ricardo Elpidio Castillo, So. Bethlehem, Pa.

Proposed by Howard Eckfeldt, W. G. Matteson, Joseph W. Richards.

Born 1889, Santiago, Cuba. 1896-1903, General education. 1903-06, High School education, Santiago, Cuba. 1906, Grad., Provincial Institute of Oriente, Cuba. B. 1910-11, English studies, Schuylkill Seminary, Reading, Pa. 1911-13, Student, Min. Engrg., Pennsylvania State College. 1906-08, D. Juan Portuondo's Private School, Santiago, Cuba. 1908-09, Public Works, Santiago, Cuba; later surveying work on own account.

Present position: Student, Lehigh Univ., Min. Engrg.

James J. Dowd, St. Louis, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1895, St. Louis, Mo. 1912, Yeatman High School, St. Louis, Mo. 1912, Mo. School of Mines, Rolla, Mo. 1914, Timber helper, Superior Copper Min. Co., Houghton, Mich. 1915, Helper, Sampler, Mill Constructor, Media Min. Co., Webb City, Mo.

Present position: Student, Missouri School of Mines.

John Joseph Doyle, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, H. A. Buehler.

Born 1893, St. Louis, Mo. 1900-07, Grades Schools. 1907-11, Central High School. 1911-12, Washington Univ., St. Louis, Mo. 1910-12 (summers), Clerk;

1913-14, Asst. Engr., Jas. Black M. & C. Co., Cleveland, O. 1914 (summer), Constr. Cost Clerk, United Railways Co., St. Louis.

Present position: Chief Clerk, Missouri Bureau of Geology and Mines, and Student, Missouri School of Mines.

Charles Barliand Gold, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1893, Montreal, Canada. 1908-12, Yeatman High School, St. Louis, Mo.

Present position: 1912 to date; Student, Missouri School of Mines.

E. Ross Housholder, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1893, Bowling Green, O. 1908-12, Grad., Bowling Green, O., High School. 1913-15, Student, Case School of Applied Science, Cleveland, O. 1915, Rodman with County Engineer, Bowling Green, O. 1915 (summer), Telautograph operator, Met. Dept., The Perfection Spring Co., Cleveland, O.

Present position: Student, Missouri School of Mines.

Joseph Jackson Krehs, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1893, Monticello, Ill. 1907-11, High School, Lake Charles, La. 1911-12, St. Marys College, Kansas.

Present position: 1912 to date; Student, Missouri School of Mines.

Charles Richardson, Toronto, Ont., Canada.

Proposed by George A. Guess, H. E. T. Haultain, Frederick C. Dyer.

Born 1894, Cleveland, O. 1900-08, Bolton School. 1908-12, Central High School, Cleveland, O. 1914, General underground work, Cobalt Lake Min. Co., Ltd. 1915, Mill sampler and work in refinery, Hollinger Hold Mines, Ltd.

Present position: Student, Univ. of Toronto.

Edgar Ude, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1892, St. Louis, Mo. 1908-12, McKinley High School, St. Louis, Mo.

Present position: Student, Missouri School of Mines.

Herman H. Vogel, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1894, St. Louis, Mo. 1908-12, McKinley High School, St. Louis, Mo. 1913, Timekeeper, Inspiration Cons. Copper Co. 1914, Asst. Chem., City Chemist's Office, St. Louis, Mo. 1915, Asst. Mine Engr., United Verde Copper Co., Jerome, Ariz.

Present position: Student, Missouri School of Mines.

Harold Edward White, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, Howard Eckfeldt, W. G. Matteson.

Born 1893, Buffalo, N. Y. 1899-1908, Grammer School, Normal School, Buffalo, N. Y. 1908-12, Lafayette High School, Buffalo, N. Y. 1913 and 1915 (summer), Draftsman, Lackawanna Steel Co.

Present position: Student, Lehigh Univ.

Change of Status—Junior Member to Member

Robert John Anderson, Rolla, Mo.

Born 1892, Cleveland, O. 1905-09, South High School, Cleveland, O. 1910-14, Case School of Applied Science, B. S. 1914-15, Asst. Supt., John Moore & Son, Cleveland, O.

Present position: 1915 to date; Instr. in Met., School of Mines and Met., Univ. of Missouri.

K. F. Hess, Morenci, Ariz.

Born 1890, Lancaster, Pa. 1913, Engrg. Dept., Ray Cons. Copper Co., Ray, Ariz. 1914, Sampling mineral property, So. Ariz. Ariz. and Nev. for private individuals and laborer and miner, United Verde Copper Co., Jerome, Ariz. 1915, Underground and churn drill sampling, Arizona and Detroit Copper Co., Morenci, Ariz.

Present position: Flotation Foreman, D. C. Concentrator.

Reed W. Hyde, Murray, Utah.

Born —. Since graduating from Columbia Univ. in 1913, have been employed at Murray Plant, American Smelt. & Ref. Co. first as assist. Chem., then as assist. to Supt.

Present position: In charge of research and experimental work, American Smelt. & Ref. Co.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Nov. 10 to Dec. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 107, April to December, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Dec. 10, 1915.

ADAMS, ARTHUR K.	Mascot Copper Co., Dos Cabezas, Ariz.
ADAMS, JOHN H.	Box 116, Winchester, Ky.
ALDRICH, TRUMAN H., JR.	1026 Glen Iris Ave., Birmingham, Ala.
ALLEN, CHESTER ARTHUR, Supt. of Construction, Oswego Milling Co.	Oswego, N. Y.
ALLEN, ROY H.	Room 204, Miners Bank Bldg., Joplin, Mo.
AMIDON, RICHARD G.	1586 Fourth St., Baker, Ore.
ANDROS, STEPHEN O.	Albuquerque, N. Mex.
AUSTIN, L. S.	251 W. 2nd North St., Salt Lake City, Utah.
BANKS, HARRY P.	715 No. Ave. 65, Garvanza Sta., Los Angeles, Cal.
CHEVRILLON, LOUIS.	23 Rue de la Paix, Paris, France.
CLARK, DOUGLAS.	Stanford Univ., Cal.
COGHELL, WILLIAM H.	Care School of Mines, Corvallis, Ore.
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 YAKE, ELMER E. "The Oaks," Springfield, Mass.

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WARNER, CHARLES M.	Care St. Joseph Lead Co., Herculaneum, Mo.
WEST, WILLIAM C.	Fort Myers, Florida.
WETHERED, ROY	Sunset, Idaho.

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Name.	Last address of Record, from which Mail has been Returned.
AUER, CHARLES I.	Estacion de Pedricena, Durango, Mex.
BARNES, BLAKESLEE	Care Arrow Engineering Co., Palmyra, Mo.
BELL, D. A. S.	136 McLaren St., Ottawa, Canada.
BLOW, JOHN J.	172 Rodney St., Brooklyn, N. Y.
BOYS, HARRY R.	Care Aurora Cons. Min. Co., Aurora, Nev.
BUNKER, CHARLES EMMONS	Promontorio, Estacion Chinacates, Durango, Mex.
BUTLER, JOHN S.	Apartado 132, San Luis Potosi, Mex.
CAMPBELL, W. C.	4 Princess St., Roodeport, Transvaal, So. Africa.
CHAMBERLAIN, JOHN R.	El Aguila Oil Co., Apartado 150, Tampico, Tamps., Mex.
CLERC, CAMILLE	92 Rue Jouffrey, Paris, France.
COLE, ROBERT J.	McKay Apts., 7th & Pike Sts., Seattle, Wash.
COLLINS, W. J.	222 Mathilda St., Pittsburgh, Pa.
COOK, PAUL RICHARDSON	4159 Grand Boulevard, Chicago, Ill.
CRARY, CHARLES N.	Kimberly, Nev.
DOWLER, HARRY P.	Penn Mary Coal Co., Heilwood, Pa.
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DUFFIN, JAMES	Box 204, Chiratera, Cananea, Sonora, Mex.
EATON, EDWIN R.	Manatawny Bessemer Ore Co., Manatawny, Pa.
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GUNTHER, C. GODFREY	Stratford, Conn.
HALLORAN, WILL	Box 13, Basin, Mont., P. O. N.
HEBBARD, AUSTIN	Box 98, Langlaagte, Transvaal, So. Africa.
HELM, J. D.	Apartado 1277, Mexico City, Mex.
HOBART, EDMUND NORRIS	Clifton, Ariz.

HOLDEN, ROY J.	Blacksburg, Va.
HYDE, JAMES M.	1841 Shattuck Ave., Berkeley, Cal.
JONES, THOMAS J.	Kyshtim Min. Wks., Perm Govt., Russia, via Petrograd.
KERR, D. GILLESPIE	Confederation Life Bldg., Toronto, Canada.
KIRKLAND, T. C., Cia.	Met. y Refinadora del Pacifico, S. A., Fundicion, Son., Me.
KISHMAN, MAURICE W.	101 E. Masonic Ave., Cripple Creek, Colo.
KITSON, HOWARD W.	Care British Columbia Copper Co., Voight's Camp, B. C., Canada.
KNOERTZER, HENRI	58 Rue Balagny, Paris, France.
KURIE, F. M.	625 I. W. Hellman Bldg., Los Angeles, Cal.
LAMB, WILLIAM H., JR.	Boyles, Ala.
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LE NOIR, FRANK H.	Douglas, Alaska.
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MILLER, FRANK BARTON	Blair, Esmeralda Co., Nev.
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NEWMAN, BRUNO	Apartado 90, Aguascalientes, Mex.
NOBS, FREDERICK W.	La Leonessa, Matagalpa, Nicaragua, C. A.
NORRIE, WILLIAM G.	Lucky Jim Zinc Mines, Three Forks, B. C., Canada.
PALMER, CORTLANDT E.	2 Rector St., New York, N. Y.
PINKHAM, W. F.	Battle Mountain, Nev.
PORTER, ROBERT S.	Care Fortifications, Culebra, Canal Zone.
PRETTYMAN, FRANK REMINGTON	Oyster Bay, L. I.
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YOUNG, CARL DEUEL	Tower City, S. D.

NECROLOGY

Date of
Election.
1906

Name.
* Wilson, John B.

Date of Decease.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Recrystallization of Cold-Worked Alpha Brass on Annealing*

BY C. H. MATHEWSON,† PH. D., NEW HAVEN, CONN., AND ARTHUR PHILLIPS,‡ M. S.,
BRIDGEPORT, CONN.

(New York Meeting, February, 1916)

DURING the past year considerable work dealing with the mechanical properties and microstructure following the anneal under uniform conditions of certain types of commercial rolled brass has been done in this laboratory. Since these experiments were conducted with the active coöperation of the Bridgeport Brass Co., involving free use of its product, and have thus furnished information which is characteristic of some of its special mixtures, full publication of the numerical relationships and comparative results encountered does not seem justifiable. Some of the more general features of this work are, however, available and it is our present purpose to use the material in question, along with some results of special tests, in discussing the characteristics of recrystallization as related to the degree of hardening by strain and the ordinary annealing variables. We desire to make acknowledgment of the many courtesies extended by Mr. Charles Ferry, Metallurgist of the Bridgeport Brass Co., and the value of his coöperation, which has greatly stimulated the study of brass in this laboratory.

A clear and systematic presentation of the principal relations between temperature and time of anneal, the changing physical properties and microstructure and the degree of previous reduction by cold-rolling, was first made, in the case of rolled brass and copper, by the French engineer, C. Grard,¹ in 1909, although much fragmentary information of this general character had appeared earlier and the main facts involved were undoubtedly known to many brass specialists through their own private research. From this work, it appears that in the cartridge mixture (67 per cent. copper, 33 per cent. zinc) the properties imparted by cold-working within the limits, 13 to 75 per cent. reduction of area by rolling

* Metallographic communication from the Hammond Laboratory of the Sheffield Scientific School, Yale University.

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¹ C. Grard: *Cuivre et Laitons à Cartouches, Révue d'Artillerie*, Feb.-Apr., 1909.

$\left(\frac{\text{Original Area} - \text{Final Area}}{\text{Original Area}} = 0.13 \text{ to } 0.75 \right)$, are not affected by annealing for a 50-min. period at any temperature below 200°C. At or in the vicinity of this temperature, metal which had been reduced 75 per cent. showed a sensible decrease in tensile strength and a corresponding increase in elongation, while metal which had been reduced 13 per cent. showed no change. An intermediate value of reduction, 56 per cent., the one which Grard evidently used in most of his tests, gave earliest evidences of change at about 250°. Thus, the effect of anneal in its incipient stages depends somewhat upon the previous treatment of the metal. As the temperature of anneal is increased, the change in properties becomes more marked in each case, until, at about 325°, the individual curves coincide, and from this point, the final condition produced by anneal is independent of the previous reduction within the limits investigated. With regard to peculiarities of inflexion, Grard distinguishes a number of zones corresponding to regions of permanent, or of changing properties in one sense or another. In accepting such conclusions, the empirical character of the work must be borne in mind. Grard has shown, as indicated above, that the degree of reduction by rolling determines the temperature range of permanency in the original strain-hardened condition. The reasons for this and the prevailing possibilities of variation will at least be indicated from a general point of view in the present paper.

In the principal set of curves, Grard adopted a period of 30 min. in which to complete the anneal at any given temperature. From his temperature-time curves it appears that in all cases the rate of change of physical properties, which is greatest in the earlier stages of the anneal, has become very slow at the expiration of the whole period. Moreover, as the temperature of anneal is raised, a decreasing fraction of the total 30-min. period suffices to produce most of the annealing effect. That this is a generally prevalent condition was shown by T. K. Rose² who, in studying the annealing properties of a variety of coinage alloys by the scleroscopic method, makes the following statement: "The annealing action at any given temperature is most rapid at first and gradually dies away, so that a state of equilibrium is not attained in any short space of time, except perhaps at high temperatures. For example, equilibrium apparently was not attained in the gold-copper, silver-copper, or nickel-copper alloys, or in pure nickel, in 16 days at 300°. At high temperatures, almost the whole result is obtained instantaneously, though a little extra softness is got by prolonging the time. A few minutes at such a temperature cannot be replaced, for practical purposes, even by hours of annealing at much lower temperatures."

This work of Grard has been widely circulated among brass metallur-

² T. K. Rose: On the Annealing of Coinage Alloys, *Journal of the Institute of Metals*, vol. viii, pp. 86-125 (1912).

gists through the medium of reprint, translation, and review. Doubtless it has supplied information of direct numerical value in some quarters, while in others it has served as a welcome guide in the development and coördination of testing data. The latter type of service appears to us to be the most desirable function of publications of this sort. The individual brass mill may be urged to develop its own data of standardization, or control, whereby all local matters of manufacturing practice, peculiarities or specialties in composition, and other individual features, may receive consideration in their proper environment, leaving broader conceptions subject to development and modification by exchange of ideas through the scientific press.

With these ideas in mind, the following notes are selected from work which we have done, with the hope that they may be of some service to those who are interested in the annealing characteristics of brass. In these notes some addition to the commonly available information on brass metallurgy has been made chiefly along the following lines:

(a) By presenting a few carefully conducted physical tests to show the general direction and magnitude of changes which are effected by heating cold-rolled strips below an effective annealing temperature.

(b) By determining the periods of time necessary to produce a measurable amount of softening in one kind of strain-hardened material at a succession of temperatures below the region of rapid effects.

(c) By emphasizing the micrographic aspects of strain-hardening and recrystallization in metal which has received both light and heavy reductions by rolling.

(d) By discussing the relations between temperature and time of anneal, degree of deformation, and structural alteration in alpha brass, with an attempt to give some general explanation of the facts involved.

(e) By showing comparisons between the ordinary physical properties and the grain size, both given as functions of the annealing temperature for a fixed period of anneal.

Changes Produced in Cold-Rolled Strips by Heating One-Half Hour at 200°C.

Bengough's* experiments on 70/30 brass, while devoted largely to the question of "burning," extend over a wide enough range to furnish a broad basis of comparison with those of Grard. Both observers have recorded a similar sequence of changes in the strength-properties after a constant period of anneal at different temperatures, except that, according to Bengough, the metal becomes several per cent. stronger after anneal at temperatures in the vicinity of 300°, while Grard, in his curves, gives no indication of an increase in tensile strength before the normal (reverse) effect, due to anneal at higher temperatures, sets in.

It may be urged that variations of this order may be expected, in

* Bengough and Hudson: Heat Treatment of Brass, *Journal of the Institute of Metals*, vol. iv, pp. 92-127 (1910).

view of unavoidable slight imperfections in the test pieces, or on account of testing errors, irrespective of any actual annealing effect. It is obviously necessary to obtain the mean values from a moderately large number of tests before reasonably certain conclusions can be drawn. We are not certain whether Bengough used several test pieces for each experiment, or relied upon a single piece to represent each anneal. In any event, it is true that, in all three series embracing low-temperature anneals, viz., copper 69.4 per cent., $\frac{3}{4}$ -in. round bar; copper 71 per cent., $\frac{3}{4}$ -in. round bar, and copper 71 per cent., $\frac{1}{8}$ -in. wire, somewhat higher tensile strength was obtained in samples annealed between 285° and 320° (different series), than in the original hard-rolled material. A critical examination of Grard's data is not possible, since the actual values are not tabulated or shown in his diagrams.

Our method of testing the point at issue was as follows: Strips of the dimensions, 0.1 by 0.5 by 8 in., were sawed from a cold-rolled sheet of 70/30 brass* which had received a dead soft anneal in the mill previous to the final reduction of 40 per cent. Five of these strips were tested in a 50,000-lb. Riehle universal testing machine in the cold-rolled condition. Another set of five was tested after annealing for a period of one-half hour at 200°. The material under present consideration cannot be heated for a half-hour period at temperatures in the neighborhood of 300°, viz., at temperatures similar to those given by Bengough, without appreciable softening and visible recrystallization (true annealing effect). One of our tests at 300° showed a softening of four points on the scleroscope scale and abundant recrystallization, as illustrated in Fig. A, Plate I. The main reason for this difference lies in the moderate initial reductions by cold-working received by Bengough's rods, as compared with the heavy initial reduction received by our strips. While Bengough does not give the reductions used in his material, the above conclusion is justified by the materially lower tensile strength and higher elongation of his bars, as compared with our strips; all tested in the cold-rolled condition.

The annealing was conducted in an electric resistance furnace of rectangular box-shaped type containing a central tube of alundum (14-in. long by 1 in. bore) closely wound with "Nichrome" ribbon cemented into place with alundum cement. To avoid rapid radiation from this heating unit, it was embedded in a mixture of granular fire clay and silica.

Two test strips were treated simultaneously (except in the annealing of the fifth strip). The strips were clamped in a vise and a hole large enough to accommodate the thermo-couple tube was drilled at one end in such manner as to bore both pieces simultaneously for a distance of approximately $\frac{1}{2}$ in. in the direction of their length. Two wire clasps held the strips in this position and served as a support for the metal when placed

* Analysis: Copper, 69.35; iron, 0.04; lead, 0.02 per cent.

in the center of the furnace tube. All bars for tests and micrographic observations were annealed in this way. After anneal, the drilled ends were cut off and micrographic observations made in the immediate vicinity of the region where the thermo-couple junction had previously rested. It is to be noted that the junction was virtually imbedded in the metal at this point.

Owing to increased radiation at the ends of the furnace, the temperature of the tube at the center was found by preliminary experiments to be about 10° higher than at the ends. This difference is small and is actually less in the bars themselves, as their high heat conductivity tends toward equalization of temperature, *i.e.*, flattens the thermal gradient.

Probably the most effective method of flattening the thermal gradient in a wire-wound tube furnace is to make use of independent heating coils around the ends which may be run at a temperature just high enough to compensate for the increased radiation at the ends. Gray⁴ has obtained ideal conditions with this type of construction. By decreasing the distance between turns from center to ends, a compensating effect may be secured, but with any given winding the desired effect will be secured only when the furnace is running at certain temperatures; above and below this range, the ends will be over- and under-compensated, respectively.

Gray's method complicates the construction of the furnace and requires independent circuits along with independent control, while the other method requires a number of different resistors to be used for different temperature ranges and there is great difficulty in properly spacing the windings. We have sought to improve the gradient without introducing another element of control and without greatly complicating the construction of the furnace. Lately, we have used furnaces in which the thickness of the heat-insulating covering around the tube (granular fire-clay) increases from the center to the ends, thus permitting more rapid loss of heat in the region of the center. In principle, this is similar to varying the space between turns, but it is far easier to accomplish, and any adjustment may be changed in a few minutes. This modification merely requires the construction of false inner sides of sheet iron which are bent to give the proper inclination from center to ends and slipped within the furnace box (made of asbestos wood). The granular fire-clay is then filled in with a scoop. It is clear that, according to this construction, the thickness of the heat-insulating covering varies from the center of the tube to the ends, but the depth is constant, *viz.*, the correction applies to the sides of the tube but not to the top and bottom. While it would be preferable to make the correction concentric with the tube, this would entail greater complication and less flexibility. These

⁴ A. W. Gray: Production of Temperature Uniformity in an Electric Furnace, *Bulletin Bureau of Standards*, vol. x, pp. 451-473 (1914).

furnaces may be taken down in a few minutes. They are capable of reducing the maximum variation of temperature along 8 in. of the central portion of the 18-in. tube from 30°C. to less than 10°C. within the range 200° to 1,000°, using a single adjustment.

In all of the annealing work, the test pieces were placed in a hot furnace and rapidly brought to the annealing temperature which was then kept constant for the desired period by adjusting the external resistance. With the present set, a preheating period of 10 min. is to be added to the annealing period of 30 min. The results are shown in Table I.

TABLE I.—*Early Effects of Annealing upon Physical Properties of Worked Brass*

No.	Treatment: Time and Temperature of Anneal	Tensile Strength in Pounds per Square Inch	Percentage Elongation in 2 in.	Hardness by Scleroscope
1	As rolled (40 per cent. reduction).....	76,829	10.0	33.2
2	As rolled (40 per cent. reduction).....	76,424	11.0	34.4
3	As rolled (40 per cent. reduction).....	76,822	11.0	33.0
4	As rolled (40 per cent. reduction).....	77,017	11.0	33.0
5	As rolled (40 per cent. reduction).....	76,723	12.5	33.2
Average		76,763	11.1	33.2
6	½ hr. 200°C.....	78,415	10.5	34.0
7	½ hr. 200°C.....	79,166	10.0	34.2
8	½ hr. 200°C.....	78,615	10.5	34.4
9	½ hr. 200°C.....	77,914	9.0	34.0
10	½ hr. 200°C.....	78,450	10.0	34.2
Average		78,512	10.0	34.2

The two sets of numerical values are consistent and seem to indicate that there is a slight but unmistakable increase in tensile strength and a corresponding decrease in elongation produced by heating at 200°C. It would seem that Bengough and Hudson's curves indicate more accurately than Grard's the true conditions brought about by heating at low temperatures.

The precise nature of the changes which take place in worked metal as a result of exposure to very moderate temperatures, or even spontaneously, is not understood. Any attempt to study these changes with the aid of the microscope is certain to fail, since the accompanying structural alterations are too minute or too indefinite to be followed. In the present case, careful comparisons between the two sets of strips were made with the aid of a $\frac{1}{2}$ -in. immersion objective without detecting any difference. Both the heated and the original strips show the typical folds, or wrinkles, which are prominent in heavily worked brass and may be seen under low power in the photomicrographs of Plate II, or under high power ($\times 1,000$) in Fig. A, Plate I, leaving out the central, recrystallized portion.

It is obvious that since no alteration of structure can be associated with these effects they must be of a mechanical nature, or they must be due to some reorganization of the particles which does not suffice to change the etching characteristics of the metal. The former interpretation is practically substantiated by the recent work of E. Heyn⁵ in which the distribution of internal strains in cold-worked metal before and after heat treatment is determined by means of special tests. This subject is handled at length and in a masterly manner by Heyn. We will merely draw attention to his observation that reheating at temperatures even below the softening point is productive of "remarkable effects toward diminishing the degree of internal strains" with the comment that similar effects in the present experiments are in all probability responsible for the differences observed in the ordinary tensile test. Heating at or above the earliest softening point would entirely remove internal strain but would also lower the tensile strength. It would seem from the experiments in hand that the frequent destructive effect of aging (season-cracking) in severely worked metal, which is undoubtedly due to relief of internal strain from some local cause, could be avoided in every case by suitable heat treatment without loss of tensile strength. The present tests show that there may actually be an increase in tensile strength associated with the partial release of strains in some rolled shapes.

Time Required at a Number of Different Temperatures between 225° and 325°C. to Produce a Drop of 3 Points in Scleroscopic Hardness in the Case of 70/30 Brass which had Received 40 Per Cent. Reduction by Rolling after a Dead Soft Anneal in the Mill

The same material was used in these tests as in those just described. A number of 1-in. pieces cut from the 8-in. strips were placed in the furnace at the given temperature and removed at intervals.

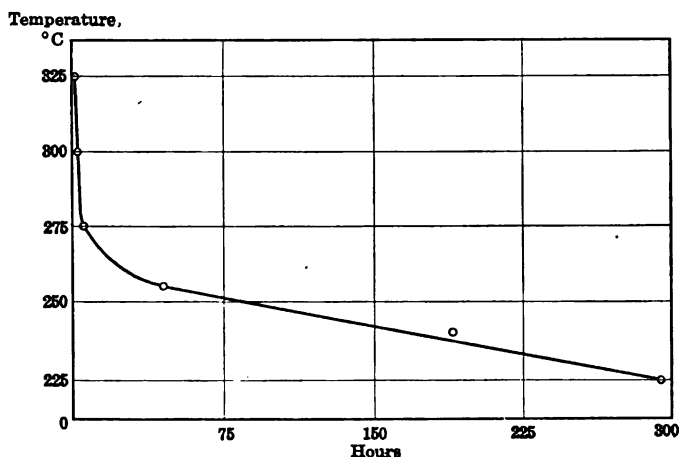
TABLE II.—*Factors Corresponding to a Drop of 3 Points in Scleroscopic Hardness after 40 Per Cent. Reduction by Cold-Rolling*

Temperature of Anneal	Time of Anneal
°C.	Hours
225	294
240	189
255	46
275	5¼
300	2½
	Minutes
325	18

⁵ E. Heyn: Internal Strains in Cold-Wrought Metals and Some Troubles Caused Thereby, *Journal of the Institute of Metals*, vol. xii, pp. 3-37 (1914).

At 425° and 375°, the strip pieces could not be brought to the furnace temperature, which required about 5 min., without losing more than the desired 3 points on the scleroscope scale. In the former case, the loss was 13 points, and in the latter, 8 points. Between 325° and 225°, the time necessary for a 3-point drop could be measured and the values obtained are given in Table II.

Fig. 1 shows the above results in graphical form. From a practical standpoint, if we are to speak of a "critical" temperature at which softening will begin in any particular mixture which has been given a stated reduction by cold-working, we must not omit certain qualifications, since the same effect will be produced at a number of different temperatures depending upon the time allowed. From the indications of this curve and from other considerations to be shown later, there is reason to believe



Time necessary to cause drop of 3 points in scleroscopic hardness.

FIG. 1.—CARTRIDGE BRASS (70/30) REDUCED 40 PER CENT. IN AREA OF SECTION BY COLD-ROLLING.

that there is a minimum temperature corresponding to a given degree of reduction at which the first softening effects may occur, but at this temperature the change proceeds very slowly, so that a period of days, or months, may be required to bring about equilibrium. The particular equilibrium involved will be discussed in another section of this paper.

In the present case, the arbitrarily chosen unit of measurable softening* is realized at a temperature as low as 225°, but only after a period of about 12½ days, while, at a temperature only 50° higher, the same effect is produced in about 5 hr., and at a temperature still higher by 50°,

* We have chosen 3 points on the scleroscope scale as the measure of appreciable softening since there can be no doubt from an experimental standpoint that the metal has softened when this indication is obtained. A drop of two points is not as easy to detect, while an indicated drop of 1 point is always open to doubt.

viz., at 325°, the effect is produced in a period of minutes. None of the values shown on the curve are equilibrium values, since further softening occurs at any given temperature on longer exposure; from the flattening of the curve, however, it is apparent that the lowest value given, viz., 225° is not far above the true equilibrium temperature for this degree of softening.

Bearing these facts in mind, it is clear that any set of curves giving the physical properties of a commercial mixture after anneal for a fixed period of time following a given reduction by cold-working requires that the stated period of anneal be rigidly adhered to within the critical range of incipient effects, in order that the results of test may be properly duplicated. At higher temperatures, when all changes are rapid, approximate equilibrium values are reached in comparatively brief periods of time and some latitude in this respect may be allowed without materially affecting the results. For example, in the case of cartridge brass (70/30±3), which has received a heavy reduction, at all temperatures above 325°C., variation of the annealing period between ½ hr. and 1 hr. will not bring about material change in properties; at somewhat lower temperatures, however, incipient effects are encountered and the time factor must be carefully established by test.

Early Stages of Recrystallization in Material which has Received a High Degree of Reduction by Cold Work

Rose observes² that the change in hardness and the recrystallization, in the case of gold, take place almost simultaneously, with a slight lag in the visible recrystallization; especially if the specimen is heated for a short time only. As far as can be inferred, his observations were all made at low magnification and, in case of a slight drop in hardness, he failed to detect recrystallization. As a result of the experiments just described, we are in possession of a large number of small pieces of metal which show different degrees of hardness varying from 33, that of the original cold-rolled metal, to 14, and these variations were brought about by heating for different periods of time at the various temperatures within the range, 225° to 325°. A sufficient number of these were carefully examined to prove beyond doubt that any positive indication of softening by the scleroscope can be detected in the form of recrystallization under the microscope. This applies only to material which has received comparatively heavy reductions, as will be pointed out more clearly in a later section.

In order to detect recrystallization of this order, microscopic examination under high-power is necessary, as the first visible elements of recrystallization are extremely minute. Before describing these effects,

² *Journal of the Institute of Metals*, vol. viii, pp. 86-125 (1912).

it may be well to refer briefly to the ordinary appearance of brass after different degrees of strain-hardening, as seen under the moderate magnifications usually adopted in brass work. A set of photomicrographs* representing the effect of cold-rolling in stages from gages 18 to 25 (B. & S.) using alpha brass containing about 64 per cent. copper, 0.2 per cent. lead and 0.04 per cent. iron is shown in Plate II. The reductions are given with the micrographs. The samples were etched with ammonia and hydrogen peroxide: The common high brass, from which these micrographs were made, and cartridge metal, which is the subject of the present experiments, both show the same deformational characteristics. It may be noted that Fig. F, Plate II represents approximately the same degree of reduction as that received by the metal used for the present tests.

In general, reductions of 5 or 10 per cent. have little or no effect on the etching properties of alpha brass. Photographs taken before and after deformation are very similar; practically the only alteration due to moderate working of the metal which may be observed in the photograph is a slight displacement of the normally rectilinear boundaries between parts of a twinned grain, so that they frequently appear curved. Some of these curved bands may be seen in Fig. B, but, more prominently, in the succeeding figures. When the reduction reaches about 20 per cent., the etching characteristics of the metal begin to change; there is a loss of contrast between differently orientated grains owing to the development of a secondary structure within the grains themselves. This is barely discernible in Fig. C, but becomes more prominent as we pass to higher reductions in the succeeding figures. At a reduction of about 30 per cent., as in Fig. D, the direction of extension in cold-working becomes apparent, in that the grains themselves show a prevailing direction of elongation.

The intergranular secondary structure due to deformation consists in the main of groups of curved or wavy lines which possess the same general direction in each grain, or homogeneous portion of a twinned grain, and increase in number and prominence with the degree of reduction. They incline to a direction at right angles to the direction of elongation and it appears that, in the squeezing of the grains between the rolls, the most destructive movements affecting the integrity and homogeneity of the grain substance have occurred chiefly along the gliding planes which, in any given grain, are nearest at right angles to the direction of elongation.

In the absence of any generally accepted term of definition, these lines may be called, simply, lines of deformation. They are the most prominent and the most characteristic lines which are produced by any deformational process. In some laboratories, the term slip bands has been dis-

* These photomicrographs were prepared by Philip Davidson in the Hammond Laboratory for the Scovill Manufacturing Co. who have kindly permitted us to use them in the present connection.

torted out of its original meaning and made to include markings of this character. It seems wiser to restrict this term to its original meaning which relates only to the visible effects produced when associated units of grain-substance move along the crystallographic gliding planes and break the plane of the polished surface. It does not appear wholly correct to make use of the term, Neumann's lines, in this connection, since these markings, which are revealed on etching and doubtless represent etching characteristics dependent upon a previous form of mechanical treatment, have been studied chiefly in connection with meteoric iron, in which they were originally discovered (Neumann, 1850), and have received an authoritative interpretation based upon crystallographic relationships in the case of iron alone.^{6, 7}

When a specimen of rolled brass showing lines of deformation is manipulated under the higher magnifications, it is seen that the black lines are shade effects primarily due to differential etching of the surface into curved elements of varying breadth and elevation. The surfaces of intersection of neighboring elements reflect the incident rays away from the line of vision producing dark bands whose breadth and quality vary with the focussing position and the angle of illumination. Where the original orientation of the grain is such as to reflect very little light to the eye, *i.e.*, in the case of a dark grain, the deformational detail may yet be seen to some extent, since part of the light reflected from the curved surfaces of intersection between bands reaches the eye. Thus, the light grains as a whole are rendered darker, while the dark grains are rendered lighter, so that the original contrasts are greatly reduced. This is apparent in the series of photomicrographs shown in Plate II.

The deformational detail of several grains may be seen in Fig. A, Plate I, under a magnification of 1,000. The recrystallized central portion of the figure need not be considered in the present connection. An acute-angled grain in the upper right-hand portion of the photograph shows a uniformly close arrangement of bands, wrinkles, or folds, as they may be variously termed. It may not be inappropriate to pattern after the term slip bands, and designate these bands, which constitute the etching record of intercrystalline deformation, *etch bands*.

The grain shown in the lower left-hand portion of the figure is marked with wider bands, while the grain just above this has reflected little light upon the plate and gives an unsatisfactory rendering of detail. In some cases, the etch bands are crossed by a second set of bands; this is seen in the extreme upper center of the photograph.

The edges of the etch bands are frequently serrated, as shown in Fig. B, Plate I. This indicates that the existing positions have been reached

⁶ Osmond, Fremont and Cartaud: *Les Modes des Deformation, R vue de Metallurgie*, i, p. 11 (1904).

⁷ Osmond and Cartaud: *The Crystallography of Iron, Journal of the Iron and Steel Institute*, iii, p. 444 (1906).

through movements of step-form, which is in accord with present conceptions of plastic deformation. It is probable that serrated edges would always be observed if adequate magnifications could be used.

While direct observations of this character cannot fail to aid the imagination in devising theories to account for the series of transformations which are effected by severe mechanical treatment followed by annealing, the actual processes of reorganization cannot be observed owing to the minute character of the participating units. A few remarks bearing upon this subject will be made in following paragraphs, along with brief reference to leading theories.

In a recrystallized structure, free from mechanical alteration, each homogeneous member or grain shows its individuality on etching by reason of its directional properties, which determine selective etching with boundary distinctions, or selective reflection with contrast effects. When the grains are strained beyond the limit of elasticity, their temperature range of existence without visible change of form is restricted. On heating to a high enough temperature, they appear to disintegrate into smaller grains. It is, however, entirely at variance with the laws of energetics for large grains to disintegrate spontaneously into smaller grains and thereby accumulate additional surface energy. Therefore, the increase in surface energy must have developed as a result of the strain, *i.e.*, in the form of inner surfaces between blocks, or groups, of the grain substance. This mechanical dislocation of the grain substance is not indicated in any way by etching except in cases of severe strain. For example, the inner surfaces which must exist after a 5 or 10 per cent. reduction by rolling, since the metal will thereafter develop a secondary structure on annealing, are not indicated in the corresponding photographs, Figs. A and B, respectively, of Plate II. A more complete illustration may be had by referring to Plates VII and VIII, the first of which represents cartridge metal which has received an 8 per cent. reduction, followed by a series of annealed structures; and the second, similar metal which has received a reduction of 12 per cent. followed by a series of annealed structures. In neither case has the rolling produced any clearly discernible dislocation within the individual grains, but, in both cases, the coarse grain structure is refined by annealing at 550°.

It is thus evident that we cannot see the inner surfaces and we can only speculate as to their nature. Since differences in orientation are ordinarily revealed on etching, it may be argued that the deformation has not brought about differences in orientation, but has merely displaced blocks of component particles along the natural gliding planes. If this is all that has occurred, it is difficult to see why any reorganization should result on annealing, since the particles are still arranged in conformity with normal crystallographic requirements, *i.e.*, the original space-lattices are preserved.

As a matter of fact, a sample of brass, which had been heated to 750° so as to develop a coarse grain and then strained so as to show numerous slip bands on the polished surface (see Fig. A, Plate III), gave little or no evidence of recrystallization after annealing at approximately 600° (see Fig. B, Plate III). The sample was photographed in the original position after annealing and, making due allowance for changes in the dimensions of the grains as a result of repolishing (in which several thousandths of an inch was removed from the surface), it is clear that the original structural outlines are generally preserved in the annealed metal.

Tammann's theory of plastic deformation⁸ is founded upon crystallographic principles and admits only of movements of translation, or of movements into twinning position, along the natural gliding planes. The occurrence of inner surfaces along the gliding planes would necessitate their subsequent removal on annealing, but no reason is apparent why this should not be effected by recoalescence in normal alignment with preservation of the original space-lattices. There would then be no recrystallization and the grain would acquire stability in its altered form.

It is not impossible that slight differences in orientation may occur without influencing the etching characteristics to a visible extent. What appears to be a single grain may, in some cases, be a close association of two grains which chance to have about the same orientation. We have frequently observed cases where the distinction between two neighboring grains was difficult to make. Thus, the deformational process may bring about surfaces of discontinuity between blocks of material within a single grain, whereby the orientation changes only slightly from block to block. There would then be opportunity for recrystallization on anneal with accentuation of visible differences of orientation, as the normal process of coalescence brings about a survival of those grains, or reorganized blocks of material, which possess the strongest individualistic tendencies and can maintain contact existence.

Plastic deformation, according to Lehmann⁹ (see also Czochralski¹⁰), results in an altered arrangement of the grain substance under the combined influence of the imposed forces and those forces which ordinarily operate within the molecular aggregate. Thus, blocks or groups of particles are forced into a new position of more or less orderly arrangement in which certain axial and other relationships are assumed. A conception of this sort would imply far-reaching reorganization of the grain substance at elevated temperature, since forms of natural stability

⁸ G. Tamann: *Zeitschr. für Elektrochemie*, vol. xviii, p. 584 (1912); also *Lehrbuch der Metallographie* (Leipzig, 1914), Chapter 1, C.

⁹ O. Lehmann: Spontane und Erzwungene Homoötropie, *Intern. Zeitschr. für Metallographie*, vol. vi, pp. 217-237 (1914).

¹⁰ Czochralski: Gegen die Translations-Hypothese, *Intern. Zeitschr. für Metallographie*, vol. vi, pp. 289-296 (1914).

(recrystallized units) would then replace the mechanically organized forms, with removal of artificial inner surfaces.

It is but a step from the above theory of displacement to the amorphous theory developed by Rosenhain¹¹ as an extension of Beilby's earlier work in this field. Instead of an orderly displacement of the affected particles, this theory would lead us to believe that truly amorphous membranes, or envelopes, are formed along the articulating surfaces. In other words, certain of the grain particles are rubbed out of alignment and given an isotropic arrangement when blocks or layers of grain-substance move against one another. Hardness and strength are supposed to be specific properties of the amorphous cementing material, thus explaining strain-hardening at ordinary temperature. The inner surfaces, in this case, are the surfaces of contact between the two molecular varieties of material and the disappearance of such surfaces at elevated temperature, owing to transformation of the unstable amorphous material into stable crystalline material with reorganization and growth of grain, is a rational development.

No adequate discussion of the merits of these theories is contemplated in these pages; it has merely seemed advisable to have the main points of the most prominent theories in mind while considering such visible effects of structural reorganization as we have been able to define in the case of brass. As previously pointed out, we cannot hope to see the early stages of this reorganization under the microscope for obvious reasons.

Turning again to the photomicrographs of rolled metal, it may be remarked that inner surfaces of the nature required by Rosenhain's theory would, in all probability, fail to show on etching owing to the extreme thinness of the amorphous membranes which Rosenhain conceives to be of molecular dimensions. When the deformation reaches a high value, as shown in the last five figures of Plate II under low power, and in the first figure of Plate I under high power, the etch bands appear. This constitutes direct evidence that the material within a given grain is broken up into groups which differ among themselves in some fundamental essential. The mere fact that there are etching distinctions indicates that there are also underlying distinctions affecting the grouping and alignment of the component particles. When these groups reorganize on annealing, the banded form is entirely lost; in other words, a band of uniform appearance under the microscope disintegrates and participates in a form of reorganization which leaves no indication of its former position. Even the smallest of those recrystallized units which can be clearly resolved at a magnification of 1,000 seem to bear no directional relationship to the preëxisting banded structure. This, in all probability, indi-

¹¹ W. Rosenhain: *Der Kristallisierte und der amorphe Zustand der Metalle, Intern. Zeitschr. für Metallographie*, vol. V, p. 65-106 (1913). A bibliography of the earlier papers by Beilby is given on p. 103 of this paper.

cates that the earliest visible stages of reorganization represent conditions which exist after considerable coalescence and rearrangement have occurred, so that the first movements away from the banded structure are obscure.

It is clear that while the etch bands probably are indicative of severe dislocation, or development of inner surfaces of marked discontinuity in the general direction of one favored set of cleavage or slip planes, the absence of inner surfaces corresponding to displacements in a crosswise direction is not necessarily indicated by the lack of markings in this direction. In this connection, we should recall the entire lack of markings in metal which has received very little reduction and yet is susceptible to recrystallization. From the facts available, it is fair to conclude that the etch bands in severely deformed metal represent the development of innumerable inner surfaces of discontinuity which are highly specialized and continuous in some prevailing direction, so as to give rise to selective etching effects.

The central area of Fig. A, Plate I, shows a number of recrystallized units which seem to have grown out of the surrounding matrix of indefinable structure. This matrix no longer contains etch bands and it seems to represent a condition intermediate between visible recrystallized units and visible strain effects. In Fig. C of the same plate, we can see where the etch bands of a large, light grain in the upper left-hand corner have broken down along a jagged boundary, to the right of which the altered and indefinable matrix occurs, along with a quantity of reorientated units. It may be remarked that the first portions of metal to recrystallize are almost always in regions of intersection between a number of grains, as is the case in Fig. C. Since recrystallization occurs at successively lower temperatures with increasing reductions, it appears that these regions are regions of maximum strain, which would, indeed, be indicated by the conflicting systems of etch bands and the oppositional nature of the forces in these regions. Figs. B and D, Plate I, taken from a piece of metal which had just begun to soften, show the earliest effects which can be distinguished under a magnification of 1,000. In Fig. B, a number of minute and not very clearly distinguishable recrystallized units may be seen in regions where the etch bands have degenerated into a matrix whose structure is not dissolved. In Fig. D, the etch bands have partially broken down, while no reorientated units are distinctly shown.

We have thus far dealt with the early stages of recrystallization in metal which has received a comparatively heavy reduction by rolling. After such reduction, recrystallization begins, as we have seen, in the more severely strained areas at low temperatures, the exact values of which in any case depend upon the degree of reduction and the time of exposure. As the temperature of anneal increases under otherwise

uniform conditions, additional etch bands break down, or granulate; the existing recrystallized units coalesce and grow; and, at no very elevated temperature, the entire preëxisting grain structure becomes replaced by a secondary, refined structure of rather uniform character. The condition which results after such a series of changes is illustrated by Fig. D, Plate X, corresponding to a reduction of 23 per cent.; an annealing temperature of 550°; and a period of 30 min.

Early Stages of Recrystallization in Material which has Received a Light Reduction

A few special experiments were made for the purpose of revealing the characteristics of recrystallization in metal which had received very light reductions. In these experiments, the reductions were not measured, but they were brought about by pressure at right angles to a polished surface; so some idea of their intensity and effects could be gathered from the changing appearance of the polished surface with its slip bands and other indications of movements in the surface region. Furthermore, in every observation, the specimen was placed in the same position on the stage of the microscope, so that the appearance of the same portion of the surface could be compared after various operations. In order to enhance the value of such comparisons, a very coarse grain was developed at the start by annealing at a high temperature. By this precaution the necessary operation of repolishing did not result in total obliteration of grains which were previously visible at the surface. It is obvious that such procedure will not enable us to refer all changes to grains previously seen on the surface, since any removal of the surface will reveal new grains at certain places, *i.e.*, where the subjacent grain boundaries were not far below the first surface. Some advantages are, nevertheless, to be derived from this method of comparison.

Space can be given to only a few of the large number of photomicrographs obtained. These illustrations, together with a brief description of the results in general will serve to characterize the recrystallization phenomena.*

The surface-effect of a moderate amount of strain after anneal at 750° is shown in Fig. A, Plate III. Slip bands are visible in all of the grains. The characteristic angular relations between slip bands where twinning planes occur, which has been illustrated in many published photomicrographs, can be seen in a number of places. The strain has been sufficiently pronounced to produce cross-slips in a number of the grains.† There has been relatively little movement of the grains with respect to

* Various brass mixtures were used in these experiments; the photomicrographs shown were obtained from a sample of admiralty mixture containing 72.53 per cent. Cu; 0.07 per cent. Fe; 0.04 per cent. Pb; and 1.06 per cent. Sn.

† Unfortunately, much of the detail of this micrograph has been lost in reproduction.

one another. As previously mentioned, we are unable to obtain positive evidence of recrystallization after annealing this specimen. Fig. B, Plate III, shows the repolished surface after anneal at 615° . Most of the grains visible in Fig. A can be identified in Fig. B. Some additional twinning bands can be seen, but this may be due to the difference of several thousandths of an inch between the two surface planes. If any crystallization has actually occurred it has only resulted in a slight increase in the number of grains in the affected region. It may be noted that the temperature of anneal, although rather high (615°), is more than 100° below the temperature of anneal before straining (750°). It is, of course, possible that a temperature of anneal in the neighborhood of this latter value would bring about distinguishable recrystallization. On the other hand, the preëxisting structure, irrespective of any alteration by strain in this experiment, might undergo some change at a temperature in the vicinity of the original anneal and we omitted the experiment on account of the uncertainties attached to it. We believe that, in the case of stresses which merely suffice to develop the ordinary slip bands but do not produce inner surfaces of another character, no recrystallization will occur on any annealing treatment purely as a result of these movements of translation. It is our impression that the numerous inner surfaces of slip, which are indicated by the close order of slip bands at the outer surface, are healed on annealing without reorientation in any grain.

The same specimen is shown in Fig. A, Plate IV, after strain of more severe character. In this case, the grains have moved considerably with respect to one another, and slip within one and the same grain has also contributed not a little to the existing surface irregularities owing to more extended movement in some parts of the grain than in others. These surface irregularities account for the blurred appearance of the surface in the micrograph; it is not possible to bring all parts into focus at once. On account of this difficulty, comparison between this and the annealed structure shown in Fig. B is more satisfactory at the microscope, where adjustments may be changed to suit any part of the surface, than through the medium of photomicrographs. It is apparent, however, particularly in the lower portion of the surface represented, that the number of grains has increased on annealing; the inner surfaces have been obliterated by reorganization into smaller grains. In the upper part of the figures, where strain was least severe, at least one grain common to both may be seen.

Another specimen, strained somewhat more severely than the last, is represented in Fig. C, Plate IV. The companion micrograph, Fig. D, shows the corresponding surface after anneal at 550° , a temperature about 65° lower than that adopted in the previous case. Here a large, banded grain near the center of the micrograph seems to correspond with one in about the same position in Fig. C, while groups of smaller grains in the

neighborhood correspond to reorganization of the much larger primary grains shown in Fig. C.

Experiments of this character were continued both in the direction of increasing the strain and decreasing the temperature of anneal. With regard to the former, it may be asserted that more severe strain decreases the size and increases the number of separate units which may be seen after a given annealing treatment. In other words, more uniform structures result when the metal is more severely strained. With regard to the latter, if the specimen is strained to about the same extent as that shown in Figs. A and C, Plate IV, it will not give evidence of recrystallization at temperatures much below 500°. The last micrograph of annealed material, viz., Fig. D, Plate IV, therefore represents very early stages of recrystallization in material which has received a very light reduction, and may be compared with Figs. A and C, Plate I, which represent the very early stages of recrystallization in material which has received a heavy reduction.

From the standpoint of potential recrystallization, the chief distinction between metal which has received a heavy reduction and metal which has received a light reduction lies in the vast difference between the number of inner surfaces which have developed in the two cases. These inner surfaces cannot remain permanently in existence under changing conditions and, as the temperature is raised, they give way to more stable configurations in accordance with equilibrium requirements which take into account their number, character, and distribution, along with the other variables encountered. Two neighboring particles with separate surface boundaries will tend to coalesce, or flow together, under the influence of forces which operate to reduce the surface area and surface energy (*i.e.*, to reduce the number of separate surfaces) to a minimum. There are oppositional forces which bring about a certain degree of firmness, or rigidity, of the grain-substance, or determine directional properties which each particle tends to retain and which resist coalescence, but all of these forces vary in intensity with the temperature and, at any given temperature, there will be a condition of equilibrium beyond which further decrease of surface area is effectively opposed by the forces in question.

This is a form of equilibrium which renders the size of the particles, or the number of inner surfaces, chiefly a function of the temperature. In order that reorganization within the grain substance may proceed at a low temperature, there must be a high order of development of inner surfaces, as in the case of metal which has received a heavy reduction. Small recrystallized units may then develop at the expense of preëxisting inner surfaces. Where the metal has received a light reduction, however, there can be no development of very small recrystallized units, since there are no inner surfaces of this order; the first units to recrystallize will be com-

paratively large, as was noticed in the case of Fig. B, Plate IV, and they will not be able to recrystallize until a comparatively high temperature is reached.

Much valuable material bearing upon the phenomena of recrystallization after strain, and the principles involved, has been contributed by Heyn,¹² with illustrations drawn chiefly from copper and iron, and by Tammann,⁸ with particular relation to his translation theory of plastic deformation. One of us¹³ has lately attempted to give an outline of the relations between the annealing variables, the degree of reduction of cold-working, and the structural characteristics in connection with the study of some ancient bronzes exhibiting a wide variety of structures. Some of this material will be used at this point for the purpose of introducing a discussion of Plates V to X which illustrate the structures obtained in the case of cartridge brass by systematic variation of the annealing temperature and the degree of reduction.

Principal Relations Between Deformational Treatment and Structural Characteristics after Anneal

The accompanying diagram (Fig. 2) will serve as a basis for the ensuing remarks. This diagram is based upon the assumption that the mean size of the new grains which form at any given temperature by coalescence of existing grains or by local breakdown of internal surfaces within the original strain-hardened grains is determined by the temperature of anneal. Thus, the size of the recrystallized grain will be some function of the annealing temperature, and the conglomerate will be composed of recrystallized grains along with unrecrystallized fragments of the mechanically altered grains. Such structures are commonly observed in strained metal which has not been annealed sufficiently to cause complete recrystallization throughout the mass, *e.g.*, Fig. B, Plate XI. The entire field is traversed by a set of lines, *ab*, *cd*, etc., which cut any vertical line into a number of segments intended to represent the mean number of recrystallized grains which would normally be counted along a unit length upon the prepared specimen after anneal at the corresponding temperature. A linear relationship is assumed in this construction, but obviously any experimentally proven relationship between grain size and annealing temperature may be depicted in similar manner.

By cold-working metal of definite grain characteristics, *e.g.*, uniformly

¹² E. Heyn: Martens-Heyn, *Materiellenkunde für den Maschinenbau*, Part II-A, Chapter IV (Berlin, 1912).

⁸ *Zeitschr. für Elektrochemie*, vol. xviii, p. 584 (1912); also *Lehrbuch der Metallographie* (Leipzig, 1914), Chapter 1, C.

¹³ C. H. Mathewson: A Metallographic Description of Some Ancient Peruvian Bronzes, *American Journal of Science*, Vol. xl, pp. 525-616 (1915).

recrystallized metal of the mean grain size which would be produced by complete anneal at T_0 , the grain structure is broken down, inner surfaces develop and new, latent grains of a lower order of size and stability are formed. As already pointed out, these secondary particles cannot be distinguished under the microscope. Etching peculiarities, diminishing contrast, lines of deformation, etc., are indeed visible when deformation has been severe, but no direct indication of the size or shape of the ultimate secondary units can be obtained. As far as direct observation goes, these new structure elements are latent only. Upon annealing, they assert a modified individuality wherein mutual readjustment and coalescence cause them to become visible units with definite orientation and the dimensions characteristic of the annealing temperature adopted.

In order that the annealing effect may be felt at a given temperature, it is clear that the mechanical destruction of the original grain must have been sufficiently pronounced to produce fragments inferior in size to the

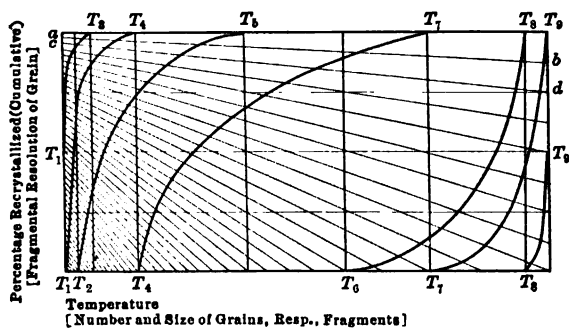


FIG. 2.

recrystallized grain characteristic of this temperature. Thus, if the T_7 grain previously cited is to recrystallize at T_6 , it must have been artificially reduced by deformation to a point where the grain size characteristic of T_6 can develop. In general, the grain fragments produced by deformational processes will vary widely in size, so that certain of them will be able to coalesce below T_6 , while others will remain unaffected at this temperature. The particular factors which apply in any given deformational process will combine to determine a curve of "*fragmental resolution of grain*"* for this particular process in which the cumulative percentage of fragments below a given size appears as a function of the size of the fragments.

* We have introduced this phraseology to aid in defining the disintegrating effect of cold-working without regard to the specific nature of the fragments, or particles, formed. Most theories freely admit that subgranular units possessing some degree of individuality are produced when the metal is stressed to the point of permanent deformation.

The curve T_4T_7 is a hypothetical curve of this sort and signifies, in the present case, that, by deformation of the original T_0 grain approximately 65 per cent. of the material has been sufficiently reduced (by internal slip) to recrystallize by the time the temperature T_6 is reached. At T_6 , then, 65 per cent. of the material will have reached the mean grain size characteristic of this temperature, while 35 per cent. will remain unannealed.

Little can be said relative to the distribution of the unannealed fragments and the recrystallized grains. In general, the former occurs in patches of irregular outline and, when present in comparatively small amount, as in the present case, cannot usually be distinguished from the latter. When present in large amount, however, the patches are large, compared with the size of the recrystallized grains, and often bear evidence of strain. Such a condition is shown in Figs. A and C, Plate I. Here large patches of metal bearing distinct lines of deformation can be seen along with small recrystallized grains.

The first formation of recrystallized grains usually occurs at the boundaries of the parent grains, since, at such localities, maximum unhomogeneity of the lines of force may be expected and, consequently, maximum fragmental resolution of grain.

As the degree of deformation increases, whether by rolling, drawing, hammering, etc., the curve of fragmental resolution recedes in the direction of maximum resolution, a specific condition for each metal or alloy beyond which further destruction of grain will bring about fracture. Recession in this direction is indicated by the set of curves, T_2T_6 , T_1T_4 , and T_1T_3 . As maximum resolution is approached, the abscissa range covered by the curve becomes narrower, since large fragments give place to smaller ones and the size of the latter cannot be reduced without limit. Ultimately we would reach a condition of uniform resolution into fragments of minimum size. It is improbable that such a condition can be realized without overworking the metal to the point of manifold fracture. Thus the limiting curve, T_1T_3 , is drawn to represent a 50 per cent. resolution into fragments of minimum size.

There is, for each metal or alloy, a minimum temperature at which recrystallization will start from a condition of maximum resolution into fragments of minimum size. This temperature is represented by T_1 in the present diagram. One-half of the metal deformed according to the curve T_1T_3 would recrystallize at T_1 . At the higher temperature, T_3 , all of the metal deformed according to the curve T_1T_3 , about 85 per cent. of the metal deformed according to T_1T_4 , about 35 per cent. of the metal deformed according to T_2T_6 , but no part of the metal deformed according to T_4T_7 would recrystallize. At T_7 , the grain of all four would be identical and would grow uniformly beyond this temperature. These curves all represent severe deformation, in that the percentage of coarse frag-

ments is low compared with the percentage of fine fragments. In the alpha brasses, such a condition occurs when the reduction by rolling or hammering is carried beyond about 15 or 20 per cent. Above 550° , provided the annealing period is not confined to a few minutes, alpha brass of definite composition will give rather uniform grain characteristics whatever the extent of previous reduction beyond the minimum value given. Hence, we may draw certain conclusions relative to heat treatment from these grain characteristics without intimate knowledge of the previous mechanical treatment.

Turning now to the effect of anneal upon metal which has received very light deformational treatment, it appears that the curves of fragmental resolution will here assume a somewhat different form, in that the percentage of coarse fragments will very likely exceed the percentage of fine fragments. This condition is represented by the curves, T_6T_8 , T_7T_9 , and T_8T_9 . Moreover, none of the fragments are likely to be very small and, accordingly, recrystallization will not start until moderately high temperatures have been attained. Recrystallization according to the curve T_6T_8 starts at T_6 , but is confined to small portions of the mass until the temperature rises toward T_8 . At the latter temperature, the entire alloy is recrystallized. According to the curve, T_8T_9 , the extremely light deformation indicated has left 50 per cent. of the original T_9 grain unaltered, while the remainder has been broken down sufficiently to cause initial recrystallization at T_8 , a temperature not greatly below the original annealing temperature. At T_9 , the metal has assumed its original grain characteristics and growth will continue normally as the temperature is further elevated. The curve, T_7T_9 , represents an intermediate condition which may easily be interpreted.

In cases of light deformation, recrystallization cannot always be detected under the microscope, since the recrystallized grains are not greatly inferior in size to the original grains. In this connection, it should be noted that a section through the conglomerate shows large and small grain sections whatever the true size of grain, since a grain may be cut at any point according to its position with regard to the cutting plane. This renders it difficult to detect recrystallization except where the new grains are considerably smaller than the original grains and are sufficiently numerous to form groups of characteristic appearance.

The curves of fragmental resolution are purely hypothetical and represent in a general way what seem to be reasonable characteristics of the ordinary deformational processes. The precise form of any curve will be determined by the nature and intensity of the deformation sustained by the metal.

It is clear from the discussion thus far that the form of a curve of fragmental resolution determines the form of a curve representing the cumulative percentage of metal recrystallized as a function of the tem-

perature. Without seeking to establish the exact relationship between these curves, it may be assumed that they are substantially identical in form, the former representing an obscure condition and the latter a visible effect. By careful counting under suitably chosen magnification, the approximate form of some of these curves may be determined. The present discussion is merely intended to convey a rational conception of deformational and recrystallization phenomena from a purely qualitative standpoint. As far as the present brasses go, it is possible to detect certain characteristic effects, such as an early or a late stage of recrystallization after severe deformation, without inquiring into the details of the deformational treatment (degree of deformation, etc.).

It may be remarked that a diagram of this sort may be drawn to represent the condition developed by anneal during a purely arbitrary period of time, or by anneal during such a period as may be necessary to bring about an equilibrium effect. The latter (ideal) condition was kept in mind throughout the foregoing discussion. The former condition would entail displacement of all recrystallization curves without affecting the general principles involved.

Owing to the difficulty of distinguishing between recrystallized and unrecrystallized units, a count of the grains in partially recrystallized conglomerates is not especially useful or significant. It is possible to obtain at first an increase and, later on, a decrease in the number of visible grains as the period of anneal at constant temperature, or as the temperature for constant period of anneal, increases. This does not signify that the individual grains first disintegrate (contrary to thermodynamical requirements) and later coalesce, but that the original strain-hardened grain fragments (each of which is counted as a single unit) are rapidly developing secondary grains in the early stages of the anneal; while later, coalescence alone constitutes the predominant factor.

Structural Changes Developed in 70/30 Cartridge Brass by Uniform Anneal at Six Different Temperatures within the Range, 350 to 800°, after Reductions of 2, 4, 8, 12, 15, and 25 Per Cent., Respectively

The material used in these tests came to us in the form of cold-rolled strips 8 in. long by 0.5 in. wide by 0.1 in. thick. They had received a reduction of six numbers, according to the B. & S. gage (50 per cent. reduction of sectional area), after the last anneal, and gave the following tests when in this condition:

Ultimate strength.....	87,750 lb. per square inch.
Reduction of area.....	38 per cent.
Elongation, 1 in.....	13 per cent.
Elongation, 3 in.....	5.3 per cent.
Hardness by scleroscope.....	36.5 per cent.

Our analysis of the material showed 70.16 per cent. of copper, 0.06 per cent. of iron, and a trace of lead.

Before rolling to the adopted values of reduction, these strips were brought to a condition of nearly maximum softness (7 to 7.5 by scleroscope) by annealing for a period of 20 min. at 800°C.

A small hand rolling mill was used in obtaining the series of six reductions. Very careful measurements of gage gave the following values: 2 to 2.3 per cent., 4 per cent., 7.7 per cent., 11.8 per cent., 15 per cent., and 24.3 to 26.6 per cent. Variations in the first case were due to the difficulty of uniformly regulating the very slight reduction desired (the thickness was reduced only 0.002 in.), while variations in the last case were due to an exceptional variation in the thickness of the original strip.

The rolled strips were then cut up into pieces about 1 in. long which were thereupon annealed at each of the temperatures, 350°, 450°, 550°, 650°, 750°, and 800°, in groups containing one piece to represent each reduction. The anneals were at constant temperature ($\pm 5^\circ$) for a period of 30 min. and the preheating period varied between 8 and 11 min. in the different cases. The following annealing record at 550° will give a fair idea of this part of the work:

Preheating Period	Time	Annealing Period	Temperature, Degrees Centigrade
11 min.....	3.52	 30 min	525, in center of furnace.
	3.53		Specimens entered.
	3.59		517, in center of group
	4.04		525, in center of group.
	4.15		527, in center of group.
	4.20		525, in center of group.
	4.34		524, in center of group.

The above average annealing temperature of 525°, as indicated by the thermocouple, corresponds to 550°, as corrected by interpolation on the calibration curve.

Photomicrographs taken from all samples, with the exception of those which received a 30-min. anneal at 800° following the cold-work, are assembled in Plates V to X. It is superfluous to represent the regular anneal at 800°, since the characteristics of this anneal are the same in all cases and the first photo of each plate, taken after reduction of the material which had previously received an anneal at 800°, will serve to illustrate the principal features of this high-temperature anneal.

Each plate shows the rolled structure after a given reduction, followed by the annealed structures developed in the treatment of this particular material. Not all of the micrographs will show the same quality of contrast or the same degree of surface excellence acquired in the preliminary operations of polishing and etching. Differences of this character are, of course, to be disregarded in making structural com-

parisons. A particular effort was made in each case to select a spot of average and typical appearance when preparing the micrographs.

Turning now to Plate V, which shows the worked and annealed structures corresponding to a trifling reduction of 2 per cent., we are unable to detect any certain evidence of recrystallization at any of the annealing temperatures. A certain number of small grain-sections are seen in all of the micrographs and will always be seen on the prepared surface of a coarse-grained conglomerate, since some of the grains will be cut in attenuated regions; there are not enough of these small sections to positively indicate recrystallization at the lower temperatures of anneal. Recrystallization would be even less evident at the higher temperatures, 650° or 750°, since, as we have indicated in the preceding section, the reorientated grains which form at these temperatures are comparatively large and would ordinarily be overlooked in the midst of the 800° conglomerate. On the basis of the generalizations already introduced (preceding section), we would expect recrystallization to begin at the higher temperatures after such a trifling amount of reduction, and the fact that very little of the hardness acquired as a result of the mechanical treatment is lost on annealing until the higher temperatures are reached (see next section) confirms this conclusion.

After a 4 per cent. reduction, the microscope is more effective in revealing the early stages of recrystallization. By examination of the micrographs shown in Plate VI, it is seen that a distinct refining of the grain has resulted from the 650° anneal. The micrograph corresponding to anneal at 450° suggests the possibility of recrystallization, but subsequent examination of the entire surface of the specimen has convinced us that, owing to an unfortunate choice of position, the grain shown in this micrograph is below the average in size. From previous considerations, we argue that recrystallization at 650° cannot be seen in Fig. E, Plate V, corresponding to a 2 per cent. reduction, because of the comparatively small number of inner surfaces which are of the proper character and dimensions to coalesce at 650°, while in Fig. E, Plate VI, corresponding to the 4 per cent. reduction and the same annealing temperature, there are a sufficient number of such inner surfaces to yield groups of small grains, after readjustment, which, along with the residual fragments of the parent grains, form a conglomerate that can readily be distinguished from the original aggregate of much coarser appearance.

It is clear that while this is a conglomerate—in the sense that it is composed of recrystallized units of normal average size for this temperature of recrystallization together with residual fragments of the original grains—an anneal under similar conditions, in the case of metal which had received a sufficiently heavy reduction to produce fragmental resolution into inner surfaces entirely disappearing at 650°, would yield, not a conglomerate in this sense, but an aggregate of grains possessing the

normal average size corresponding to 650° . The latter type of structure is represented by Fig. E, Plate X, and by Fig. E, Plate XI, which correspond to anneal at 650° , following reductions of 23 and 50 per cent., respectively. From a practical standpoint, to produce uniformity of structure (and uniformity of properties) at a given temperature of anneal, the metal should have received a certain minimum degree of reduction, which, in the present case, is in the neighborhood of 20 per cent. Other deductions of this sort may be made on the basis of the accompanying photomicrographs.

For obvious reasons, the first positive indication of recrystallization in a set of micrographs showing changes of structure as we proceed from one temperature of anneal to another does not represent the earliest actual recrystallization. There is an apparent lag, owing to our inability to distinguish properly a few recrystallized units from the rest of the aggregate when these units are not very small in comparison with the parent grains, *i.e.*, when the reduction has been light. In the case of heavy reductions, this difficulty is not a serious one. We have already mentioned in an earlier section of this paper that a drop of one or two points on the scleroscope scale can be detected in the form of recrystallization by examination under high power when the reduction has been severe. With a reduction as low as 2 per cent. (Plate V) the total drop of about three points throughout the entire annealing range (350° to 800°) could not be detected in this form, as we have lately seen. It is clear that no advantage would be derived from the use of higher powers in this connection, since there are no minute units to be seen.

It is not necessary to give a detailed discussion of the other plates of the set (Plates VII to X). It will be noticed that, as the degree of reduction increases, recrystallization can be distinguished at successively lower temperatures and the first recrystallized units become smaller and smaller, as is required by the theory. Table III gives a summary in this respect.

TABLE III

Percentage Reduction	Temperature of Earliest Visible Recrystallization at 85 Diameters after $\frac{1}{4}$ -hr. Anneal
2	
4	650°
8	550°
12	550°
15	450°
25	350°
40	275° to 300° ($\times 1,000$)

Recrystallization at 350° after a reduction of 25 per cent. is not clearly shown in the corresponding micrograph (Fig. B, Plate X). A few groups

of very small units may be seen† when the micrograph is examined carefully. High-power examination leaves no doubt of the actual recrystallization, although only a small part of the metal has become affected. Cartridge brass of this composition, which has been reduced 50 per cent. and then annealed for 30 min. at this temperature, shows abundant recrystallization with residual patches of strain-hardened metal. Fig. B, Plate XI, is thoroughly representative of this condition, although it relates directly to a mixture of somewhat different composition.

The characteristics of rolled structures have already been illustrated (Plate II) and briefly described. Additional illustration is afforded by Fig. A of each of the Plates V to X inclusive. Increased curvature of the twinning bands is noticeable as the reduction increases, while the etch bands first attain prominence in Fig. A, Plate X, corresponding to a reduction of 25 per cent.

Distribution of the Total Drop in Scleroscopic Hardness in Each of the Above Series According to 100° Temperature Increments of Anneal

The small pieces of metal from which the photomicrographs described in the preceding section were made furnished suitable material for scleroscopic tests. After removing the top and bottom surface layers by light grinding with moderately fine emery, from 10 to 20 readings were taken for each piece. Variations of one, or even two points were commonly encountered in the same piece, particularly in the harder pieces. The averages, along with calculations of the comparative drop in hardness from temperature to temperature, are given in Table IV.

TABLE IV

Reduction by Rolling	2 Per Cent.	4 Per Cent.	8 Per Cent.	12 Per Cent.	15 Per Cent.	25 Per Cent.	50 Per Cent. **
Scler. No. after rolling	9.7	12.7	15.2	17.3	21.5	27.4	38.2
Scler. No. after 350° anneal	9.6	12.7	14.6	16.6	19.5	23.6	16.8
Drop between 0° and 350°, per cent.* ..	3.8	0.0	7.5	7.0	14.6	19.4	68.6
Scler. No. after 450° anneal	9.5	12.0	13.2	13.6	13.0	13.7	13.5
Drop between 350° and 450°, per cent.* ..	3.8	13.5	17.5	30.0	47.4	50.5	10.6
Scler. No. after 550° anneal	9.3	10.9	11.1	9.8	9.1	11.8	10.5
Drop between 450° and 550°, per cent.* ..	7.7	21.2	26.2	38.0	28.5	9.7	9.6
Scler. No. after 650° anneal	8.7	9.3	8.6	8.3	8.6	10.2	8.0
Drop between 550° and 650°, per cent.* ..	23.1	30.8	31.2	15.0	3.6	8.2	8.0
Scler. No. after 750° anneal	7.7	8.0	7.3	7.5	8.3	8.2	7.0
Drop between 650° and 750°, per cent.* ..	38.5	25.0	16.2	8.0	2.2	10.2	3.2
Scler. No. after 800° anneal	7.1	7.5	7.2	7.3	7.8	7.8	
Drop between 750° and 800°, per cent.* ..	23.1	9.6	1.2	2.0	3.6	2.0	
Total drop, points.....	2.6	5.2	8.0	10.0	13.7	19.6	31.2

* Given as a percentage of the total drop.

** The original cold-rolled material was used for these tests. It differs from the other material in that no preliminary anneal at 800° was introduced; pieces were cut from the original strips, which had received 50 per cent. reduction in the mill, and were given the successive anneals under the uniform conditions observed in all of the work.

† This observation applies to the original micrograph; owing to loss of detail in reproduction it may not be possible to see these units.

The proportion of the total drop in hardness which is realized during each temperature stage of the anneal is plotted in Fig. 3 for each series (by reductions), as a function of the temperature which completes the given stage.

We cannot assert that the values given in Table IV and the correspond-

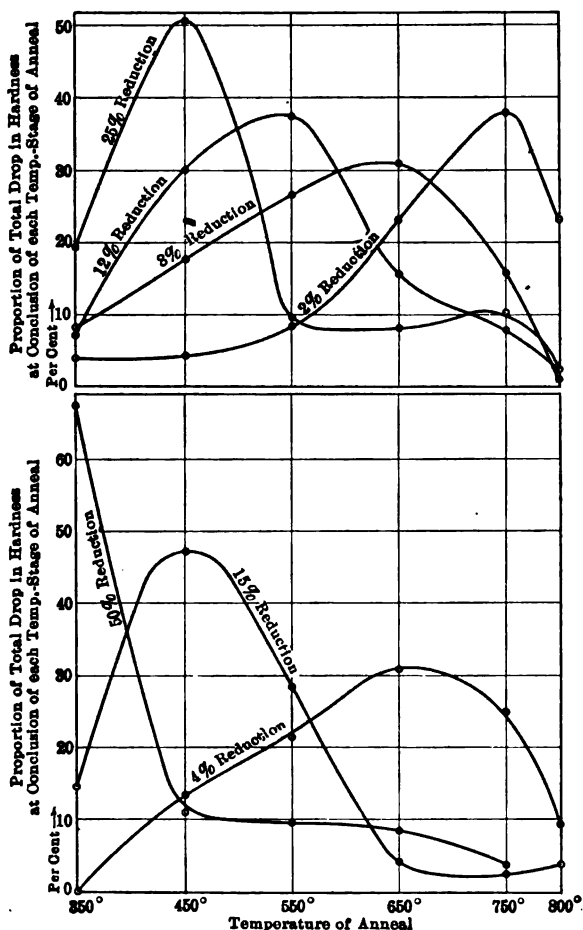


FIG. 3.—DISTRIBUTION OF HARDNESS DROP ACCORDING TO SUCCESSIVE TEMPERATURE STAGES OF ANNEAL—CARTRIDGE BRASS (70/30). THE PERCENTAGE REDUCTION IS INDICATED ON EACH CURVE.

ing curves of Fig. 3 are accurate to any high degree. Many of the drops were in the neighborhood of one point, according to our measurements. Our experience indicates that it is not reasonable to expect anything more than approximate hardness indications of this order. Thus, the entire curve corresponding to a 2 per cent. reduction is of an approximate nature and the same applies to portions of the other curves which corre-

spond to slight changes in hardness. The principal indications of these curves are, however, of value and may be freely used to supplement the information gained from the study of the photomicrographs.

First of all, we observe that there is a maximum on each curve which unquestionably points to particularly active recrystallization in this region and bears favorable comparison with the corresponding structural evidence. For example, the maximum on the curve representing a 25 per cent. reduction occurs in the neighborhood of 450° , which is also true of the curve representing a 15 per cent. reduction, and we have previously observed that the earliest prominent development of recrystallized units occurred at this temperature in both cases (Fig. C, Plate X and Fig. C, Plate IX). The curve representing a 12 per cent. reduction possesses a maximum in the neighborhood of 550° and we first observed recrystallization at this point (Fig. D, Plate VIII). In the curves representing 8 and 4 per cent. reductions, the comparatively flat maxima are located (probably) to the left and to the right of 650° , respectively, while we were able to detect recrystallization at 550° in the former (Fig. D, Plate VII) and at 650° in the latter (Fig. E, Plate VI). The curve representing the lowest reduction of the series, 2 per cent., rises to a maximum at the highest temperature of all, viz., about 750° , but, for reasons which have been set forth, we could not detect recrystallization at this, or any other temperature.

Assuming that the drop in hardness is due chiefly to recrystallization, a restricted temperature range of rapid drop would correspond to a restricted temperature range of active recrystallization, while a wider range would correspond to more gradual recrystallization over a wider range of temperature. According to this interpretation, the curves which are based upon the higher reductions of 50, 25, and 15 per cent. and the curve which is based upon the minimum reduction of 2 per cent., all four possessing comparatively sharp maxima, represent recrystallization within a comparatively limited range of temperature, while the curves which are based upon the medium reductions of 12, 8, and 4 per cent., all three possessing comparatively flat maxima, represent recrystallization throughout the greater part of the entire range shown in the diagram. Since we can clearly distinguish only the most pronounced effects by direct observation, the above deduction is not easily verified, at least with the sole aid of the material available at the present time. Aside from all questions as to the value of the experimental results involved, this is a perfectly logical deduction from a theoretical standpoint. It signifies, in this connection: (a) that the lowest reduction develops a small number of widely separated inner surfaces which resolve the original grain into comparatively few fragmental units and, consequently, break down, for the most part, at a temperature not far below the temperature of equilibrium of the normal grain; (b) that the medium reductions develop a large number of

inner surfaces which vary considerably in magnitude and begin to break down at a temperature far below the temperature of equilibrium of the normal grain, but continue in this process up to, or nearly up to, the above temperature; and (c) that the heavy reductions develop an extremely large number of inner surfaces of uniformly small size which break down at low temperature throughout a limited range. Considerations of this sort were formulated on preceding pages and idealized in Fig. 2. The curves of fragmental resolution, T_8T_9 , T_4T_7 , and TT_4 , shown in this diagram, correspond to the respective conditions (a), (b), and (c), specified above.

The question naturally arises whether the relative drop in hardness from temperature to temperature indicates the relative quantity of metal which has recrystallized in the respective interval, or is possessed of some modified significance. Since the rebound of the scleroscope hammer depends not only upon the amount of penetration at the point, but upon the elastic properties of the metal as well, it is necessary, in order that the different readings may possess a strictly comparative value, that several ill-defined properties vary uniformly as the ultimate testing results on the metal vary. Even if this condition is realized, a drop of a given number of points along different sections of the empirical scale, will not necessarily correspond to the same specific change in properties, although, within a limited reading range this difficulty must largely disappear. Therefore, we must proceed with caution in using the indications of the scleroscope as a quantitative measure of any structural change within the metal, such as the destruction of inner surfaces, or of any associated property, such as elastic limit, or ultimate strength. The validity of all possible relationships of this sort must be tested by experiment. Any simple conversion factor will, as a rule, apply only throughout a limited range, while satisfactory calculations over a wider range may sometimes be possible with the aid of formulas (often complicated) based upon an analysis of the graphical relationships encountered. Compare, in this connection, papers by Shepard and Porter¹⁴ and by Shepard and Summers¹⁵ on hardness tests of cold-rolled steel and brass, respectively.

Aside from all difficulties in the interpretation of results which originate in the choice, or application, of the testing instrument, we must distinguish between effects which are due solely to changes in the size of grain, i.e., to removal of inner surfaces between natural grains, and those which are due to actual recrystallization, or removal of inner surfaces of strain.

It is not a simple matter to test the effect of grain size upon hardness without introducing the element of strain, since great variations of grain

¹⁴ Shepard and Porter: Hardness Tests of Cold-Rolled Steel, *American Machinist*, vol. xlii, pp. 277-278 (1915).

¹⁵ Shepard and Summers: Hardness Tests of Cold-Rolled Brass (to appear in the *American Machinist*).

size cannot ordinarily be produced without first straining the metal and then annealing it. Such variations as may be produced by varying the rate of cooling in casting the metal (with subsequent annealing to insure homogeneity of composition), do not greatly influence the hardness. Those who have had occasion to test a variety of cast alpha brasses are aware that variations in hardness by the scleroscope never amount to more than a few points in metal of the same composition. George V. Caesar, in the Hammond Laboratory, has carefully tested the scleroscopic hardness of samples of pure copper and of the same grade of copper containing small quantities of cuprous oxide. In the annealed condition after rolling, the samples of pure copper show materially coarser grain than the samples of pure copper containing oxide, but the difference in hardness is not greater than would ordinarily correspond to the addition of the oxide. For example, the pure metal as cast gave a hardness number of 4.2, while metal containing 0.05 per cent. of oxygen, as cast, gave a number of 6. The difference of 1.8 points is due, principally, to the presence of oxide, as the grain, in the two cases, is at least roughly comparable in size. After a reduction of 66 $\frac{2}{3}$ per cent. followed by anneal at 925°, the pure metal gave a hardness number of 5, while the metal containing oxide gave a number of 6.4.

Figs. D and F, Plate X, of the brass series may be used to illustrate the considerable difference in grain size in the last two cases. The hardness difference of 1.4 points appears to bear little relation to the difference in grain size, since it is comparable with the difference observed in the case of the cast metal (1.8 points). In making use of these comparisons, we assume that, owing to the uniform temperature of anneal, actual recrystallization has occurred to the same extent in both kinds of material, but that in one case the grain has grown without hindrance to normal size, while in the other case the oxide has interfered mechanically with coalescence and thus reduced the size of grain. On this basis, we conclude that changes in grain size alone are not accompanied by considerable changes in hardness.

Turning again to the curves of Fig. 3, it may be safely asserted that each curve represents a summation of two curves; one corresponding to relatively slight drops which are due to growth of grain without regard to recrystallization from the strained condition, and the other corresponding to the more pronounced drops which are due to removal of the inner surfaces of strain. When the latter effect is principally confined to a limited range of temperature, some differentiation between the two effects is possible, as in the case of the four curves representing 50, 25, 15, and 12 per cent. reductions, respectively. In these curves, the pronounced effects which occur in the vicinity of the maxima and which are combination effects, as suggested above, are superseded by comparatively slight effects in the higher temperature regions which are due mainly to growth of grain. It is clear that we cannot sharply define the point at

which summation of the two effects first occurs in the case of any curve. Where the maximum is sharp, as in the case of the 50, 25 and 15 per cent. reductions, some approximation of this sort may be made and we have already used these indications to aid in interpreting the general characteristics of recrystallization (p. 2549). The use of such curves in a quantitative sense is an entirely different matter, and would involve too many assumptions to yield results of much value. For example, in the case of the curve representing a 25 per cent. reduction, we might interpret the change in direction at 550° to indicate the conclusion of recrystallization (from a condition of strain) at this point. We cannot be sure that there is no further recrystallization, *i.e.*, between 550 and 600° , since the last stages in the disappearance of inner surfaces may be too gradual to show an end point of this character on the curve. Assuming that recrystallization ceases at 550° , however, we might use the drops in hardness from temperature to temperature up to this point to indicate the actual proportions of the metal which recrystallize in the respective interval. This involves assumptions of proportionality as affecting the general indications of the instrument (see p. 2550), as well as similar assumptions covering the relative changes in hardness due to growth of grain throughout the intervals in question. It also involves a fundamental assumption that a unit drop in hardness corresponds to disappearance of inner surfaces (recrystallization) in a unit quantity of material, which can only be approximately correct since the inner surfaces are removed progressively from one stage of the anneal to another in the order of increasing size, or decreasing concentration, per unit volume. If, however, we plot the drops in hardness from temperature to temperature as cumulative percentages of the total drop up to a limiting point in the above sense, a curve is obtained which, under the above assumptions, gives some idea of the amount of material which has recrystallized at each successive temperature of anneal. In the case of the 25 per cent. reduction, the amount at 350° is 24 per cent.; at 450° it is 87 per cent.; and at 550° it is 100 per cent. The curve, T_2T_5 , of Fig. 2 very nearly corresponds to these figures if we assign temperature values of 335° , 360° , 415° , and 550° to T_2 , T_3 , T_4 , and T_5 , respectively. Other curves derived in the same way assume positions which are consistent with the general arrangement shown in Fig. 2. Owing to the approximate nature of the numerical relationships shown, very little of practical value would be gained by introducing a diagram based entirely upon the present experiments. We will merely say, in this connection, that these experiments give material support to the general theory of recrystallization discussed on pp. 2539 to 2543.

Owing to the fact that the scleroscope "measures a very complex function of different properties, into which one particular kind of hardness

merely enters as one factor"¹⁶ and therefore gives readings which, when put to any particular use, must be held in close association with the primary conditions of experiment (chemical composition, etc.), this instrument has been freely criticised and often rejected when it would have given satisfaction if used with proper discrimination. Our experience with brass samples representing every condition of strain-hardening and recrystallization has led to the conclusion that the scleroscopic test is quite as effective as any other ordinary test in detecting slight changes in temper (as ordinarily understood in brass mills), while it is infinitely superior to any other test in point of convenience and rapidity. As compared with the tensile test, a difference of one point on the scale is roughly equivalent to a difference of 1,000 lb. per square inch (50 lb. actual load on a section 0.1 by 0.5 in.) of ultimate strength in strain-hardened alpha brass. It is obvious that a number of tensile tests on carefully prepared test pieces are necessary in order to detect differences of this order, while the uncertain skin effect and possible imperfections in the test piece combine to render the results open to suspicion. Slight differences in temper from point to point are clearly revealed by the scleroscope and there is no difficulty in obtaining an average which fairly represents the condition of the metal. With regard to the micrographic method, it is apparent, from previous explanations, that slight relief of strain due to recrystallization can be detected only when very small recrystallized units are formed. The scleroscope indicates such effects whatever the previous condition of strain or the nature of the prevailing recrystallization. The Brinell ball test and the Ludwig cone test are no more sensitive than the scleroscopic test and their only advantage lies in the fact that the respective results are more easily recalculated in terms of other properties.

It is interesting to note that Rose² has defended the scleroscopic method in the face of considerable criticism. We can but consider it fortunate that his instructive studies on annealing were based upon this convenient but highly sensitive method.

*Comparison of Ordinary Physical Properties and Grain Size
in Samples of Alpha Brass after a 30-Min. Period of
Anneal at Different Temperatures*

We have made careful measurements of the number of grains per unit area of section as a function of the temperature of anneal (constant period) in various brass mixtures. The heavy-faced curve shown in Fig. 4 is representative of these measurements and applies to a mixture containing

¹⁶ W. Rosenhain: Study of Physical Metallurgy (London, 1914), Chapter X, p. 223.

² *Journal of the Institute of Metals*, vol. viii, pp. 86-125 (1912).

66.65 per cent. of copper, 0.30 per cent. of lead, and 0.08 per cent. of iron. For comparison, curves showing the hardness (by scleroscope), tensile strength, elongation, and reduction of area are given in the same figure. All of the corresponding data may be found in Table V. Representative photomicrographs are assembled in Plate XI.

TABLE V (a)

Nature of Anneal			Structural Detail							
Temperature, °C.	Time in Furnace, Min.	Time at Annealing Temperature, Min.	Magnification shown	Magnification of Count	No. Grains per Linear 2¼ In.		Average No. Grains per Linear Inch at 85 Diameters		N N'	No. Grains per Square Inch at 85 Diameters
					Direction of Rolling	Direction Perpendicular to Rolling	Direction of Rolling (N)	Reverse Direction (N')		
350	45	30	85							
450	45	30	85	119	35, 34, 33, 35, 33	41, 40, 37, 42, 36	17.40	19.90	0.87	346.2
550	45	30	85	85	26, 26, 27, 25, 26	29, 28, 28, 27, 26	9.40	10.20	0.92	95.9
650	45	30	85	17	39, 38, 36, 38, 37	39, 40, 41, 38, 39	2.76	2.84	0.98	7.8
750	41	30	85	17	27, 28, 28, 29, 27	28, 29, 26, 29, 30	2.04	2.06	0.99	4.2

TABLE V (b)

Nature of Anneal			Physical Tests							
Temperature, °C.	Time in Furnace, Min.	Time at Annealing Temperature, Min.	Initial Area	Final Area	Per Cent. Red. Area	Elongation in 1 in. Per Cent.	Elongation in 3 in. Per Cent.	Breaking Strength, Lb. per Sq. in.	Ultimate Strength, Lb. per Sq. in.	Scleroscope Number
350	45	30	0.4905×0.102 =0.0500	0.350×0.065 =0.0228	54.4	52	38	2,890	57,800	19.5
450	45	30	0.511×0.100 =0.0511	0.340×0.060 =0.0204	60.0	61	48	2,730	53,425	16.3
550	45	30	0.508×0.101 =0.0513	0.330×0.060 =0.0198	61.4	71	58	2,515	49,025	12.8
650	45	30	0.5145×0.1005 =0.0517	0.330×0.065 =0.0215	58.4	78	68	2,320	44,874	9.5
750	41	30	0.494×0.102 =0.0504	0.0335×0.065 =0.0218	56.7	77	67	2,130	42,261	7.5
Unannealed			0.4935×0.102 =0.0503	0.0410×0.080 =0.0328	34.8	15	6	4,070	80,919	34.5

The original material was given a final reduction of 50 per cent. by rolling in the mill. Its grain characteristics (Fig. A, Plate XI) indicate that it had been annealed in the neighborhood of 650° previous to this reduction. Physical tests on the strain-hardened metal are given in Table V (b).

Laboratory anneals were conducted as described on pp. 2524-5. In

Table V (a), the time during which the temperature, as measured in the drilled test pieces, remained sensibly constant at the annealing temperature is given for each set of two bars; also the total time in the furnace.

The number of grains per unit area was calculated from counting data. In counting, three arbitrary standards of magnification were adopted, viz., 17, 85, and 119 diameters (ratio, 1:5:7). For general purposes, each annealed specimen was photographed at 85 diameters. Not all such photomicrographs are suitable for use in counting, owing to the relatively small number of grains shown in the highest anneals

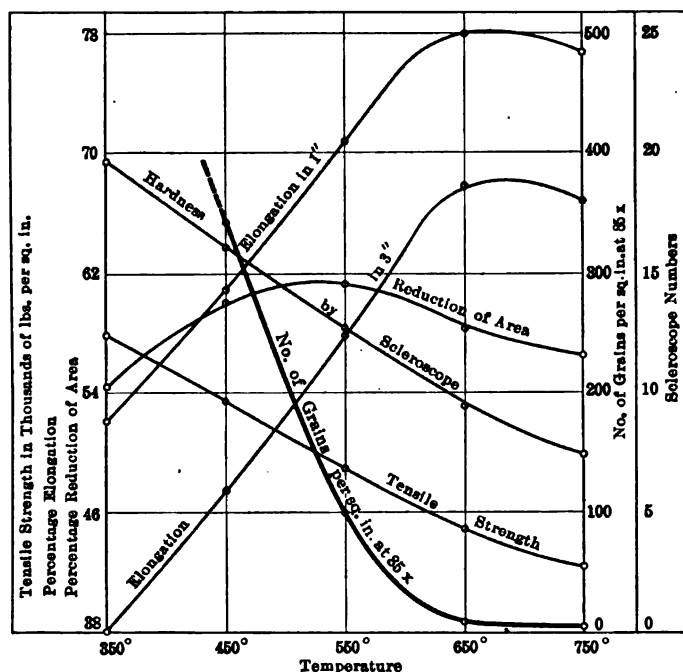


FIG. 4.—PHYSICAL PROPERTIES AND GRAIN SIZE AFTER 30-MIN. PERIOD OF ANNEALING. PREVIOUS REDUCTION, 50 PER CENT. BRASS MIXTURE CONTAINING 66.65 PER CENT. COPPER; 0.30 PER CENT. LEAD; AND 0.08 PER CENT. IRON.

and the extremely large number shown in the lowest anneals. It was deemed advisable to include at least 25 grains in each linear count and, by proper choice of one of the three above magnifications, this could be done throughout the entire series of high and low brasses. No attempt was made to count the grains developed as a result of any anneal below 450°C. Plate XI shows that below this temperature the annealed structure is made up of very minute recrystallized units along with coarse unrecrystallized fragments of the original strain-hardened grains. Aside from the mechanical difficulty of counting in such cases, we would hardly be able to make consistent use of the resulting data. The microscope

may be used most advantageously in the case of extremely low-temperature anneals to indicate approximately how much of the strain-hardened metal has been altered by recrystallization.

We may say, in general, that the effective counting range of 450° to 750° (or higher) lies within the "zone de revenu," according to Grard,¹ or the temperature region in which a slow recovery of the extreme properties conferred by anneal is effected. Recrystallization has already spread throughout the grain substance at lower temperatures and the present changes in properties are due to growth of grain alone, or to this in combination with disappearance of the grosser inner surfaces of strain (see discussion in the previous section). This range covers the ordinary annealing treatment of brass in a practical sense.

The method of counting was as follows: Five parallel equidistant lines were ruled in one direction and five in a direction at right angles to this upon a clear gelatine-coated glass plate. This plate was then placed upon the photomicrograph and counts were made along the lines for a uniform distance of $2\frac{3}{4}$ in. From the five counts in each direction, the average number of grains per inch in the direction of rolling and perpendicular to the direction of rolling was calculated. In many cases, the occurrence of twin-crystals adds to the difficulty of counting. In general, where twins are observed, the whole polygonal section of the twinned grain, embracing two or more bands, is more comparable in size to the majority of single grains than the separate sections of the twin would be. Hence, twinned grains are counted as a single grain. This is altogether the logical method, as the material is continuous in a twinned grain, i.e., there is no possibility of any (intercrystalline) impurity occurring along a twinning plane. Undoubtedly, in many cases, where the magnification is not well chosen, or the etching is somewhat below standard, or where the whole twin appears half-tone and thus shows its variations only imperfectly, it will be difficult to distinguish between twins and single individuals. On the ground of expediency, since about 300 grains are counted upon each photomicrograph, it is not advisable to spend a large amount of time in studying each doubtful case. For the same reason, it is not feasible to inquire too closely into the manner of arrangement of the grains; whether, owing to an irregular interlocking arrangement, the same grain is counted twice upon the same line, etc. In the case of low-temperature anneals, the larger grains may often represent fragments of the original grain-substance, containing within their boundaries smaller, recrystallized grains. The present method does not take proper account of such conditions. There is, of course, no conceivable method of counting the unaltered and the recrystallized areas separately, since one cannot always be distinguished from the other.

¹ *Revue d'Artillerie*, Feb.-Apr., 1909.

Relative to the general significance of the count, we can conceive of no ready method which will give the actual size of the grains, since they are cut by any plane into large and small sections as determined by their shape and orientation. As Gulliver remarks,¹⁷ when an etched area is measured and divided by the number of grains counted, the result gives only the "average area of a number of co-planar sections." It is, however, true that such procedure gives correct information as to the quadratic degree of dispersion, or the degree of discontinuity at the given location, and if a sufficiently large number of grains are counted in material of the requisite homogeneity the resulting magnitude is scarcely less definite in physical significance than a magnitude describing the actual number of grains per cubic unit of material. It is, of course, incorrect to base calculations of this sort upon measurements in one direction, since there may be a prevailing extension of grain growth in some particular direction owing to mechanical treatment or some obscure cause.

The counting data given in Table V (a) includes (columns 6 and 7) the actual number of grains counted along each line, (columns 8 and 9) averages calculated from these counts and reduced to the standard magnification of 85 diameters and the standard linear unit of 1 in., (column 10) ratios of the number of grains in the direction of rolling to the number in the reverse direction, and (column 11) the calculated number of grains per square inch at the standard magnification. To obtain the actual number of grains per square inch of etched surface, the latter value should be divided by 85×85 .

The ratios of the number of grains in the direction of rolling to the number in the reverse direction are not far from unity in the case of every anneal from which counts were obtained. This indicates that the grains in the annealed structures are equiaxed, which would naturally be expected in the event of complete recrystallization. In the case of the 450° anneal, this ratio, 0.87, is far enough below unity to give rather positive indication that the axial relationships developed in rolling have not been completely removed by anneal at this low temperature, i.e., that the material has not completely recrystallized.

On the basis of the counts, we can distinguish with ease between the four anneals at 100° intervals when suitable magnifications are used in counting. After recalculating the results in terms of number of grains per square inch, the differences between the 450° and 550° anneals, and the 550° and 650° anneals are more marked than between the 650° and 750° anneals. In other words, the number of grains per square inch increases rapidly in the early stages, but more slowly as the temperature increases. From a practical standpoint, one would be inclined to divide the entire range into four stages; one in the neighborhood of 350° giving partial recrystallization with barely distinguishable grains (at low magni-

¹⁷ G. H. Gulliver: *Metallic Alloys* (London, 1913), Chapter VII, p. 228.

fication), another in the neighborhood of 450° giving a count ranging from, say, 300 to 400 grains per square inch at 85 diameters, another in the neighborhood of 550° giving a count ranging from 50 to 100, and another in the neighborhood of 700° giving a count ranging from 5 to 10 grains. Other stages would be recognized as intermediate. This procedure, with general observations relative to homogeneity, etc., would suffice to characterize the annealing treatment.

In connection with the above statements, it may be added that some latitude must be allowed in interpreting the count, since there are counting errors amounting to at least 10 per cent. of the value obtained where the troublesome twinned grains occur in quantity and, aside from all difficulties associated with one and the same material, a second batch of metal, close to the first in composition and treated as far as possible in the same way may give somewhat different results. The following comparison will illustrate this:

The present lot of metal containing 66.65 per cent. copper, 0.30 per cent. lead, and 0.08 per cent. iron, rolled to 50 per cent. reduction after anneal at about 650° in the mill gave a count of 95.9 grains per square inch at 85 diameters and a tensile strength of 49,025 lb. per square inch after a 30-min. period of anneal at 550° .

Another lot of metal containing 66.56 per cent. copper, 0.26 per cent. lead, and 0.08 per cent. iron, rolled to 50 per cent. reduction after anneal somewhat above 650° in the mill (estimated) gave a count of 67.2 grains per square inch at 85 diameters and a tensile strength of 47,100 lb. per square inch after a 30-min. period of anneal at 550° .

There is a difference of about 30 per cent. in the number of grains per square inch and of about 4 per cent. in the tensile strength. It appears that, with due allowance for errors in counting, this method is more sensitive in detecting variations in strength properties than the tensile tests themselves.

With regard to the influence of the previous degree of deformation on the properties after anneal, both Rose¹ and Grard² showed that the properties investigated, viz., scleroscopic hardness in the first case and tensile strength, elongation, etc., in the second, were independent of the degree of (severe) deformation, provided the temperature of anneal was above a certain (low) value. In an earlier section (p. 2546) we have shown that structural identity results from anneal at 650° , in the case of cartridge brass, when the previous reduction by rolling was not less than 20 per cent. From a practical standpoint, this amount of reduction will give about the same properties after anneal at 550° as a reduction of 50 per cent., although appreciable differences in the size of grain are encountered. Table VI illustrates this statement.

¹ *Journal of the Institute of Metals*, vol. viii, pp. 86-125 (1912).

² *Révue d'Artillerie*, Feb.-Apr. (1909).

TABLE VI.—*Summary of Properties after Half-Hour Anneal at 550°—
Brass Containing 66.56 Per Cent. Copper, 0.26 Per Cent. Lead,
0.08 Per Cent. Iron*

Initial Reduction, Per Cent.	Elongation in 1 in., Per Cent.	Reduction of Area, Per Cent.	Ultimate Strength, Pounds Per Square Inch	Sclero- scope	Grains per Square Inch at 85 Diameters
20	72	59.5	46,739	10	55.5
50	73	62.0	47,100	10	67.2

Here again, the grain count furnishes the best means of distinction between two similar kinds of material.

Turning now to Fig. 4, in which the physical properties and the number of grains per unit area in the present series are shown as functions of the annealing temperature, we note that the inflexional characteristics of the curves which represent physical properties are not consistently traceable to similar characteristics affecting the curve of grain counts. In other words, the grain size is not the only factor which determines the value of any given property. Some points of similarity may, however, be observed. Thus, the curves representing percentage elongation rise rapidly in the temperature region which corresponds to a rapid fall on the grain-count curve and reach maxima in the region which corresponds to the most gradual changes in grain size. The curves representing hardness and tensile strength bear no close relation to the grain-count curve, although there is a slight falling off in the temperature-rate of change toward the end of these curves which corresponds to a marked falling off in the temperature-rate of change on the grain-count curve. On the other hand, the curve representing reduction of area of section in the tensile test shows a maximum in the temperature region which corresponds to rapid change along the grain-count curve. The maximum reduction of area almost invariably occurs at a lower temperature than the maximum elongation, although the relative positions of these points, as well as the character of the maxima (whether sharp or flat) are generally influenced by both soluble and insoluble impurities. The highest combined values of reduction of area and elongation are obtained in this mixture by anneal at a temperature in the neighborhood of 650°. This corresponds to a count of about 8 grains per square inch at 85 diameters, which value may be taken as the controlling factor in developing these properties.

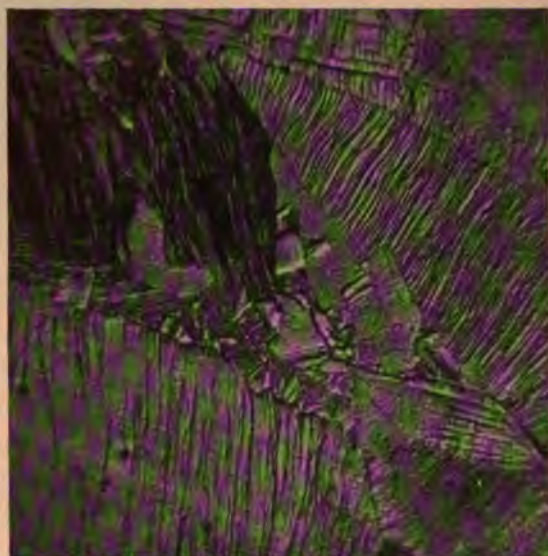


FIG. A.—From sample which had softened $2\frac{1}{2}$ points after 10 min. at 325°C . $\times 1,000$.



FIG. B.—Softening of 1 point with rise of temperature to 325°C . $\times 1,000$.



FIG. C.—Softening of 4 points after 30 min. at 300 to 325°C . $\times 500$.



FIG. D.—Softening of 1 point as in Fig. B. $\times 1,000$.

PLATE I.—EARLY STAGES OF RECRYSTALLIZATION IN CARTRIDGE BRASS (COPPER, 69.35; IRON, 0.04; LEAD, 0.03 PER CENT.) AFTER 40 PER CENT. REDUCTION BY COLD ROLLING. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



FIG. A.—5 per cent. reduction.
Gage 18½.

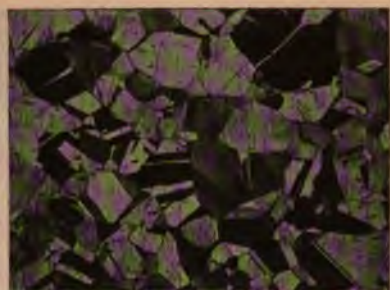


FIG. B.—10 per cent. reduction.
Gage 19.



FIG. C.—20 per cent. reduction.
Gage 20.



FIG. D.—29 per cent. reduction.
Gage 21.



FIG. E.—37 per cent. reduction.
Gage 22.



FIG. F.—43.5 per cent. reduction.
Gage 23.



FIG. G.—50 per cent. reduction.
Gage 24.



FIG. H.—55 per cent. reduction.
Gage 25.

PLATE II.—COMMON HIGH BRASS REDUCED IN STAGES FROM GAGE 18 TO GAGE 25 AFTER SOFT ANNEAL IN THE MILL. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 72$.



FIG. A.—Strained after anneal at 750° following 50 per cent. reduction.



FIG. B.—Annealed 15 min. at 615° following treatment given above.

PLATE III.—ADMIRALTY MIXTURE (COPPER, 72.53; IRON, 0.07; LEAD, 0.04; TIN, 1.06 PER CENT.). ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—Strained after anneal at 750°.



FIG. B.—Annealed 15 min. at 615° following treatment of Fig. A.



FIG. C.—Strained after anneal at 775°.



FIG. D.—Annealed 20 min. at 550° following treatment of Fig. C.

PLATE IV.—ADMIRALTY MIXTURE (COPPER, 72.53; IRON, 0.07; LEAD, 0.04; TIN, 1.06 PER CENT.). ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—2 per cent. reduction.



FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.



FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE V.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. EFFECT OF ANNEAL AFTER 2 PER CENT. REDUCTION BY ROLLING. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—4 per cent. reduction.



FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.



FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE VI.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 4 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—8 per cent. reduction.



FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.



FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE VII.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 8 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—12 per cent. reduction.



FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.

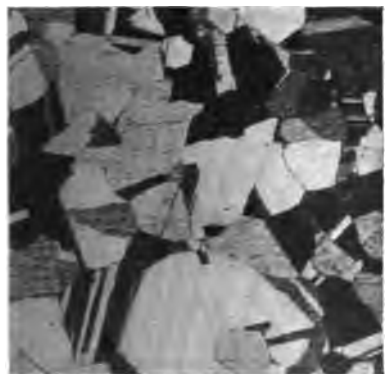


FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE VIII.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 12 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.

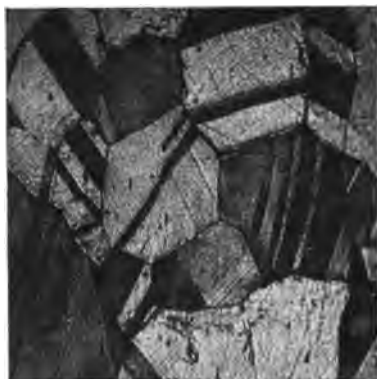


FIG. A.—15 per cent. reduction.



FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.



FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE IX.—CARTRIDGE METAL (COPPER 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 15 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—23 per cent. reduction

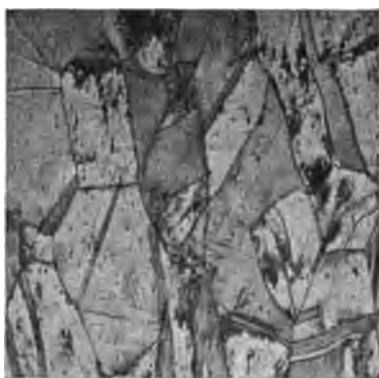


FIG. B.—30 min. at 350°C.



FIG. C.—30 min. at 450°C.

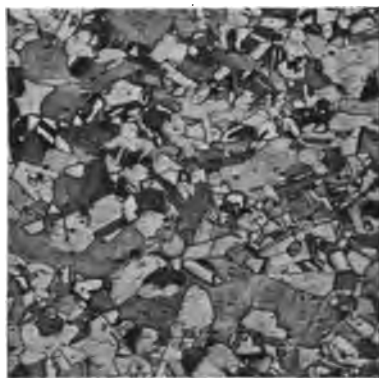


FIG. D.—30 min. at 550°C.

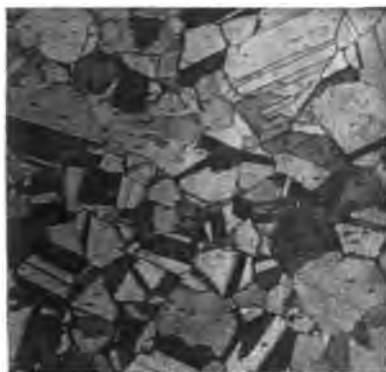


FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE X.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 23 PER CENT REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. at 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.



FIG. A.—50 per cent. reduction.

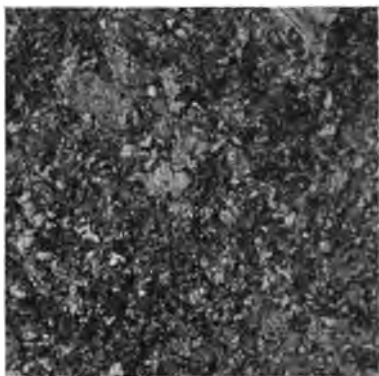


FIG. B.—30 min. at 350°C.

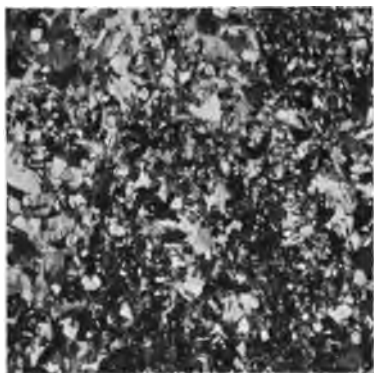


FIG. C.—30 min. at 450°C.



FIG. D.—30 min. at 550°C.



FIG. E.—30 min. at 650°C.



FIG. F.—30 min. at 750°C.

PLATE XI.—ALPHA BRASS (COPPER, 66.65; IRON, 0.08; LEAD, 0.30 PER CENT.). EFFECT OF ANNEAL AFTER 50 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED SOFT IN THE MILL BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. $\times 85$.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Underground Mining Methods of Utah Copper Co.*

BY THOMAS S. CARNAHAN, B. S.,† BINGHAM CANYON, UTAH

(New York Meeting, February, 1916)

THE mining property of the Utah Copper Co. is situated in the West Mountain mining district, Salt Lake County, Utah, in the Oquirrh Range of mountains.

GEOLOGY

In a general way the rock formation of the district consists of a series of beds of quartzite and limestone intruded by a body of monzonite porphyry roughly elliptical in shape, with an east-west axis over a mile in length and a north-south axis of about 3,000 ft.

This porphyry intrusion, accompanied by strong mineralizing action and fracturing, resulted in the formation of orebodies in the adjacent sedimentary rocks, and was itself sufficiently mineralized to make it one vast orebody. The Utah Copper Co.'s mining property comprises within its boundaries practically the entire outcrop of the monzonite mass, of a commercial grade.

GENESIS OF PORPHYRY ORE

Following an intricate system of fracturing, mineral solutions circulated freely through the porphyry, depositing small quantities of copper and iron, and resulting in a considerable silicification of the monzonite. The quantity of copper originally deposited was undoubtedly too small to have ever given the porphyry a commercial value, had not secondary enrichment, due to the leaching of the copper from the surface of the mass and re-deposition in the sulphide zone, been active over a long period of years.

A large portion of the leached material has probably been removed by erosion, but there still remains a blanket of leached and oxidized porphyry of varying thickness covering the sulphide ore, known as capping. Over certain sections of the orebody, this capping contains commercial quantities of copper carbonates, but most of it contains little or no copper.

* Originally presented at the annual meeting of the Utah Section, Salt Lake City, Oct. 4, 1915.

† Mine Engineer, Utah Copper Co.

To Jan. 1, 1915, a total of 377,690,000 tons of ore had been developed, of which 342,500,000 tons averaging 1.45 per cent. copper still remained to be mined. The average thickness of the developed ore was 465 ft., while the layer of capping covering the ore averaged 115 ft. Further development will undoubtedly show an increase in the average thickness of the ore, with a corresponding increase in the tonnage of developed ore.

UNDERGROUND MINING AUXILIARY TO STEAM-SHOVEL OPERATIONS

As is generally known the Utah Copper mine is primarily a steam-shovel operation, and it will perhaps surprise many that up to April, 1914, a considerable tonnage of ore was obtained by underground mining methods.

During the early years of steam-shovel mining the amount of ore available was naturally limited, since most of the shovels were working in capping, and it was necessary to stope a large tonnage underground in order to keep the mills at Garfield running at capacity.

During the 3-year period from 1911 to 1913 inclusive, a total of 102,719 ft. of drifts, raises, etc., was driven on the property. Most of this development served the double purpose of proving the shape and value of the orebody, and providing the necessary openings for stoping operations. The output of ore from underground operations in these 3 years amounted to 3,071,719 dry tons, of which 247,280 tons came from development and the rest from stopes.

SHRINKAGE STOPING SYSTEM ADOPTED

Realizing that underground mining was to be but an incident in the mining of the orebody as a whole, a system of stoping was adopted which would not affect adversely future steam-shovel operations. In order to fulfill this requirement it was essential that the surface should not be caved, that no large openings be left unfilled, and that the capping should not be mixed with the ore.

The system as finally adopted and successfully operated, consisted in starting stopes on three separate levels or tunnels. The first of these tunnels, at an elevation of 6,733 ft. driven 7 by 7.5 ft. in the clear, was the main or motor-haulage level. The second, at an elevation of 6,883 ft. or 150 ft. vertically above the main level, was equipped for hand tramming only, so all drifts, crosscuts, etc., were driven 5.5 by 6.5 ft. in the clear. The third, at an elevation of 6,983 ft., or 100 ft. vertically above the second, was also a hand-tramming level and driven 5.5 by 6.5 ft. in the clear. These three levels were connected by many manways and raises for dropping the ore from the upper tunnels to the motor-haulage level. An underground shaft centrally located, equipped with a com-

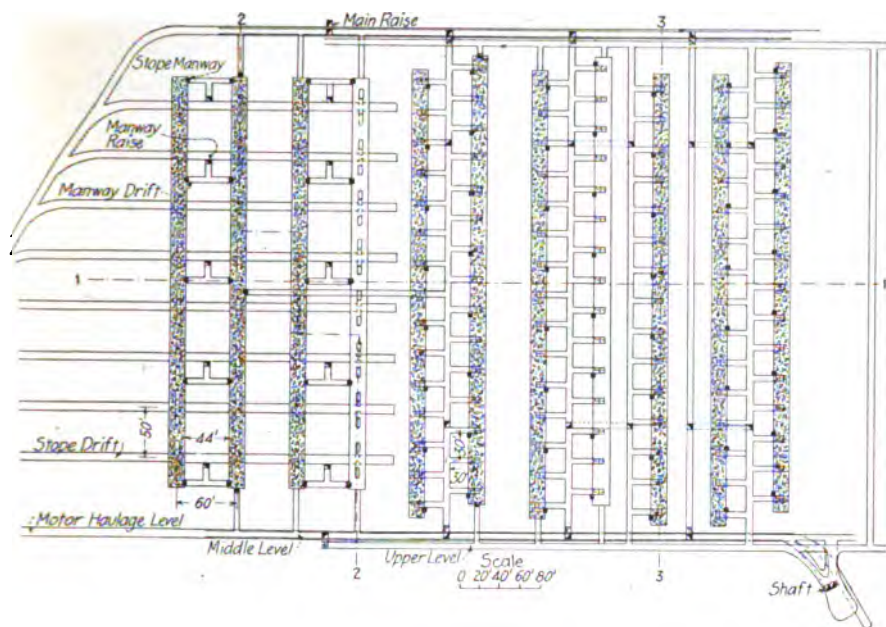


FIG. 1.—PLAN OF WORKINGS FOR ONE BLOCK OF STOPES.

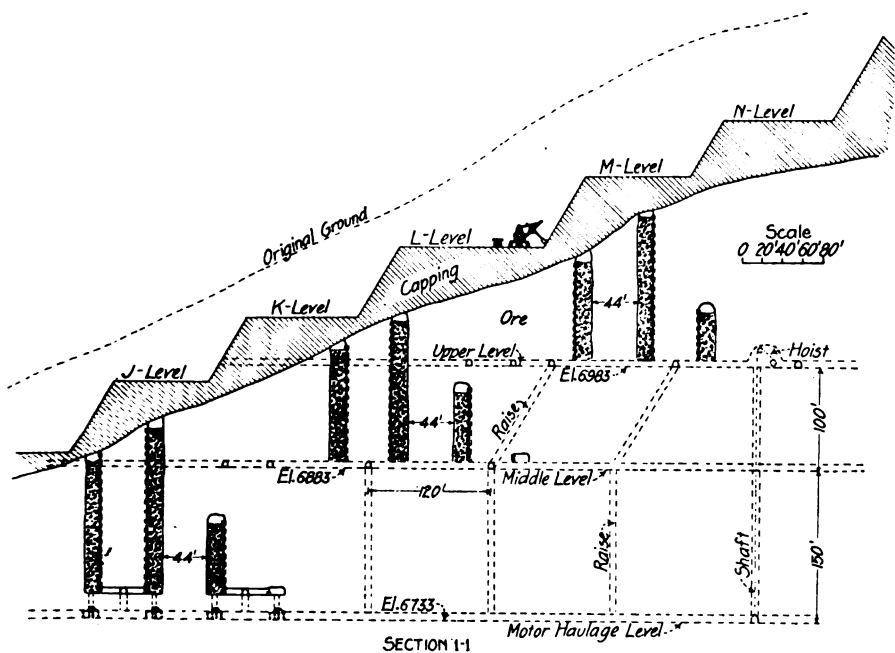


FIG. 2.—SECTION SHOWING RELATIVE POSITION OF STOPES TO STEAM-SHOVEL LEVELS.

pressed-air hoist, was used to hoist supplies from the motor-haulage level to the upper levels, but no ore was handled through it. Each level had one or more surface connections, affording good natural ventilation to all parts of the workings (Figs. 1 and 2).

OREBODY WORKED ON THREE LEVELS

By means of these three levels, the orebody was divided into blocks for stoping, the central block, 500 ft. wide (see Fig. 1), was bounded on the main level by two parallel motor drifts, and on the upper levels by two main parallel tramming drifts, directly over those on the haulage level. At intervals of 120 ft. along these motor drifts, raises 5 by 6 ft. were put up on a 55° pitch to the level above and afterward extended to the third level.

In order to make the stopes as safe as possible, to minimize the amount of timber required, and to leave substantial walls for the safety of future steam-shovel operations, it was decided that the standard width of stope should be 16 ft., with 44-ft. pillars between stopes.

The method of starting stopes on the motor-haulage level was somewhat different than on the upper levels although the size of stopes and pillars was the same.

Main or Motor-Haulage Level

Motor drifts were driven on the motor-haulage level, spaced at 50-ft. centers, and parallel to the main drifts forming the boundaries of the block. At intervals of 60 ft. along these drifts the surveyor marked the center of the stopes as the drifts were driven; if the ground required timbering the tunnel sets were spaced to be suitable for stope-chute sets later on. After the stope-chute sets were completed, a man with a stoping machine drilled both sides directly over the chutes, nearly horizontally and on the center line of the stope, to form a pocket at this point. The next round on each side pointed strongly upward, and from that point on, the raise was extended on a 60° pitch until the face was 31 ft. vertically above the top of rail. The stoping machine was then taken out, and a No. 9 Leyner machine set up near the top of the raise, and a drift started each way. These drifts were run horizontally following the center line of the stope, and were made 8 by 8 ft. Since their maximum length from any ore chute was only 18 ft., little shoveling was necessary to get the ore to the chute. After this drift was completed for the full length of the stope, the Leyner machine took 4 ft. off each side of the drift, to bring the stope to the standard width of 16 ft. The ore broken in all this work was drawn from the chutes, so that when this stage of operation was completed, an excavation 450 ft. long, 16 ft. wide and 8 ft. high was ready for stoping.

Chute timbers were constructed as follows: Three tunnel sets of 12 by 12-in. timbers were set up, spaced at 5½-ft. centers. Posts 8 ft. long

were set in hitches cut in the floor deep enough to make the bottoms of the caps 7 ft. above the top of rail. Caps were cut 10 ft. long, so as to extend 6 in. beyond the side of each post, and blocked tightly against the walls. Planks, 2 by 12 in. by 7 ft. long, nailed under the cap, acted as spreaders for the posts, but no sills were used. The sides were lagged with 2 by 12-in. planks. Enough ground was then broken above the tunnel sets to make room for a short set. The posts for this set were cut 3 ft. 9 in. long of 12 by 12-in. timber. Only two short sets were put in, one on each side of the chute mouth, the top and sides being lagged with split round poles (see Fig. 3).

Manway Raises and Drifts.—In alternating drifts, that is, at intervals of 100 ft., manway raises were put up in the pillars, midway between the stopes. The raises were started in offsets 6 ft. from the center of the track, and driven on a 50° pitch. They were made 4 by 6 ft. and divided

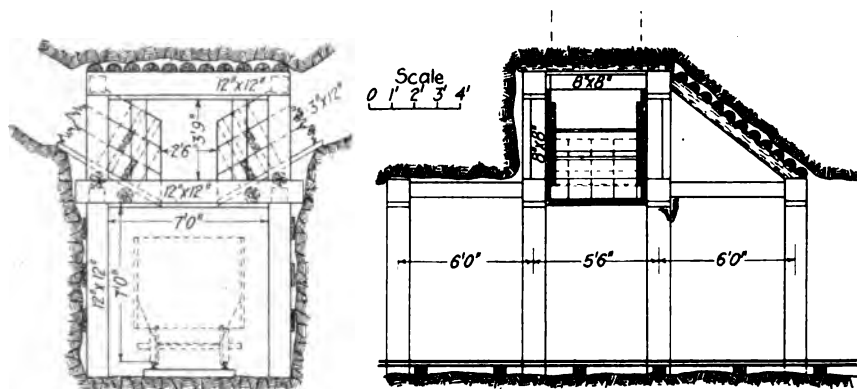


FIG. 3.—STOPE CHUTE TIMBERS ON MOTOR-HAULAGE LEVEL.

into a chute and a manway by means of stulls and 3 by 12-in. planks. The manways were equipped with ladders, and the chutes equipped with gates, so that the ore broken in the raises could be loaded direct into motor cars. When the manways reached an elevation 31 ft. vertically above the top of rail, manway drifts were started both ways at right angles to the raises, or parallel to the motor drifts. The bottoms of these drifts were 25 ft. above the top of rail, or on a level with the bottom of the stopes. These drifts, at 19 ft. from the manways, broke into the sides of the stopes. At the junction of the drifts with the stopes, 8 by 8-in. sills 8 ft. long were laid 3 ft. in the stope and 5 ft. in the drift and 6 by 6-in. cribbing built upon them, dapped 1 in. so as to leave a space of 4 in. between cribbing. The crib timbers were 4 ft. long, making the manways 3 by 3 ft. in the clear. The cribbing on the drift side was left off for the first 5 ft. to form an entrance into the manway. The manway timbers projected 2 ft. into the stope and 2 ft. into the pillar, thus placing them

in solid ground on part of three sides. After the manways were thus completed to within a few feet of the back of the stope, and equipped with air lines, the stope was ready for active stoping.

Stoping Operations.—Usually, in working a stope, a crew consisting of a machine man and a pick man worked from each manway, so that ordinarily five machines were working in each main-level stope. Stopping was done on but one shift a day, so the same crew was responsible for conditions at a given manway. The pickman trimmed down the back of the stope, so as to make it safe for the machine man. In a shift of 8 hr., each stope crew was expected to put in from 14 to 20 holes, 6 ft. deep, depending on the ground, and load and fire them. A round of 15 holes ordinarily broke 150 tons. The motor crew on the following shift pulled about 75 tons, representing the swell of the ore, so that the stope crew

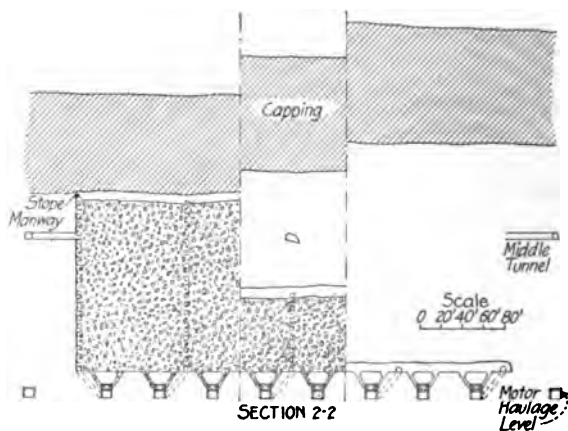


FIG. 4.—SECTION SHOWING THREE STAGES OF STOPES FROM THE MOTOR-HAULAGE LEVEL.

would have sufficient room to work on top of the broken ore. The ideal condition for efficiency and safety in a stope was to keep the top of the broken ore within 9 ft. of the back of the stope. With five machines in a stope, a separate crew of crib men was employed to build the manways, and keep them at all times well above the level of broken ore in the stope. The stopes from the main tunnel were carried to capping, or abandoned 10 ft. below the second level, if the ground above the second level had already been stoped. As soon as a stope was abandoned, the chutes were nailed up or otherwise obstructed, so that no ore could be pulled from them (Fig. 4).

Upper Levels

As already mentioned the upper levels consist of main drifts directly above the motor drifts forming the limits of the stope blocks. Crosscuts were driven every 120 ft. at right angles to the main drifts, that is, paral-

nel to the center line of the stopes. These were driven on a 1 per cent. grade from each main drift, meeting in the center of the block. Each crosscut was connected to the main-haulage level by two raises which came out in the floor of the crosscut 100 ft. from each of the main drifts. On both sides of these crosscuts, stope drifts were driven, at 30-ft. intervals, for a length of 30 ft., or 11 ft. beyond the center line of the stopes. They were then enlarged sufficiently to accommodate the timbers for stope chutes.

Chute Timbering.—The chute timbering consisted of three tunnel sets spaced at $4\frac{1}{2}$ -ft. centers, except in every other drift where an additional set was put in at $3\frac{1}{2}$ -ft. centers to accommodate the manway. Tunnel sets were of 10 by 10-in. timbers, with posts cut so as to leave 7 ft. between

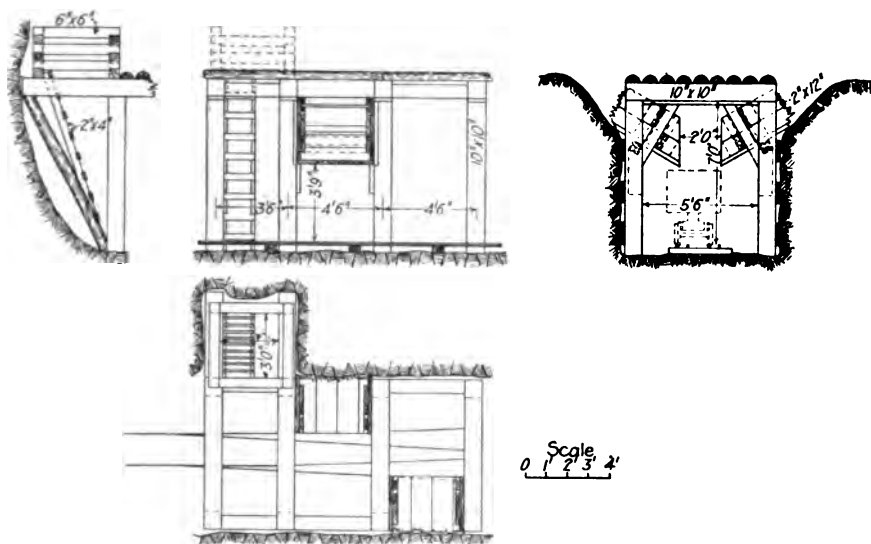


FIG. 5.—STOPE CHUTE TIMBERS ON HAND-TRAMMING LEVELS.

top of rail and bottom of cap. Collar braces were of 8 by 8-in. timbers, or round poles not less than 7 in. in diameter. Caps for manway sets were 11 ft. long to form the sill for manway cribbing. The chute lip was 3 ft. 9 in. above the top of rail and was made of 3 by 12-in. plank. Chutes were equipped with two gates so as to control the flow of ore easily. Top and sides were lagged with 2 by 12-in. plank, or split round poles (see Fig. 5).

Stope Preparation.—After the stope-chute timbers were in place, the ground above the timbers was drilled with a stope machine; but before blasting, the top was lagged with short split round poles, so that the ore could be loaded direct into a car without shoveling. An 8 by 8-ft. drift was then started on top of the chute timbers with the center line of the

stope as one side of the drift, and extended the full length of the stope. Later this drift was widened to 16 ft.

Manways.—Manways were put up in every second stope drift built of the same dimensions and material as the manways from the main level, and carried to the third level or to capping if struck before the third level was reached. Where the distance to capping was greater than 100 ft. but less than 200 ft., connection was made with the stope on the third level, the old manways abandoned, and new ones started from the third level. Where the ore thickness was greater than 200 ft. above the second level, stopes were started from the third level, run to capping, and abandoned before the stopes from the second level were started.

ORDER OF WORKING STOPES

In a given block it was the custom to have the stopes on the top level worked out considerably in advance of the stopes started from the middle

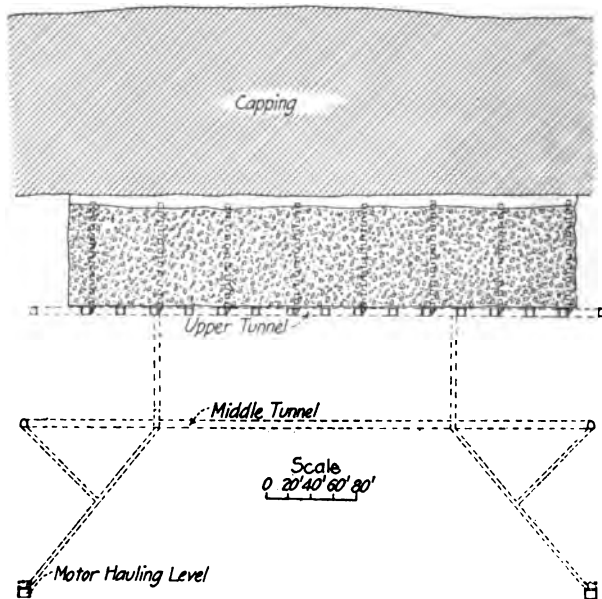


FIG. 6—SECTION SHOWING A STOPE STARTED FROM THE UPPER LEVEL, AND RAISES THROUGH WHICH THE ORE IS DROPPED TO THE MOTOR-HAULAGE LEVEL.

level. In this way the upper stopes were abandoned before the stopes from below began to disturb the tramming drifts and stope manways. This was necessary also to avoid cutting off the raises through which the ore was dumped from the upper to the main-haulage level, before the upper stopes were worked out. For the same reasons the stopes started

from the middle level were kept well in advance of those started from the main level (Fig. 6).

HAND TRAMMING ON UPPER LEVELS

On the middle and upper levels all ore was hand trammed from the stope chutes to the nearest raise. The raises were so spaced that the maximum length of tram was 150 ft., and the average 90 ft. for all stope ore. Tracks for hand tramming were 18-in. gage, laid with 16-lb. rails and a grade of 1 per cent. in favor of the loaded cars. The stope cross-cuts were double-tracked for short distances near the raises, to permit several cars to run to the same raise without delaying or interfering with each other. Tram cars measured 2 by 2.5 by 4.6 ft. in the clear, and held about 1 ton of porphyry ore. Two trammers were used on a car, and the average tonnage trammed by each crew from a stope was about 65 tons per shift. A tallyman, stationed at each active stope, credited each car dumped to the proper crew.

Data from an Average Monthly Statement

Month of August, 1912.

Tonnage from stopes.....	112,990	
Tonnage from development.....	8,026	
Distribution of Labor		Number per Day
<i>Stopes:</i>		
Machine men.....	41	
Pickmen and helpers.....	41	
Trammers.....	87	
Cribmen (stope manways).....	11	
Timbermen (stope chutes).....	11	191
<i>Development Work:</i>		
Machine men.....	28	
Muckers.....	32	
Timbermen.....	4	64
<i>General:</i>		
Trackmen.....	6	
Timbermen.....	4	
Motor crews.....	12	
Foremen and shift bosses.....	8	
Engineers and samplers.....	4	
Miscellaneous.....	29	63
<i>Surface Men:</i>		
Blacksmiths, etc.....	5	
Gravity tramway.....	20	
Common labor.....	6	
Miscellaneous.....	9	40
Total.....	358	
Average daily tonnage per man.....	10.9 tons	

GENERAL COSTS

During the 3-year period, 1911 to 1913 inclusive, 2,824,439 dry tons of ore was drawn from stopes at a cost of 56.6c. per ton loaded into railroad cars at the ore bin. In addition, 102,719 ft. of development work yielded 247,280 dry tons at a cost of \$4.95 per foot of development or \$2.05 per ton of ore. As the development was carried on entirely in average grade of ore, it about paid for itself from the ore broken. Counting every item of expense, the cost of producing a ton of ore from all sources by underground mining averaged 68.7c. By the system in use, with 16-ft. stopes and 44-ft. pillars, less than 27 per cent. of the ground developed was actually stoped, and as the stopes were full of broken ore when abandoned, the actual production represented less than 50 per cent. of the ore broken, or about 13 per cent. of the ore blocked out by the development work. The cost of production would have been considerably less, if the pillars could have been mined and the broken ore in the stopes pulled. Man-way drifts and raises, and stope drifts were charged direct to stoping, all other drifts and raises being charged to development.

Cost of Development Work

Motor-haulage drifts (7 by 7.5 ft. in the clear); No. 9 water Leyner used (progress 5 ft. per round):	
	Per Foot
Drilling and mucking.....	\$3.50
Powder, caps and fuse.....	1.25
Compressed air.....	0.30
Miscellaneous.....	0.55
Total.....	\$5.60
Timbering (where necessary).....	1.61
Total.....	\$7.21
Tramming drifts (5.5 by 6.5 ft. in the clear) 2.5-in. piston machines used (progress 3.8 ft. per round):	
Drilling and mucking.....	\$2.25
Powder, caps and fuse.....	0.68
Compressed air.....	0.21
Miscellaneous.....	0.47
Total.....	\$3.61
Timbering (where necessary).....	1.35
Total.....	\$4.96
Raises (4 by 5 ft.); hammer machines used, (progress 5 ft. per round):	
Drilling and tramming.....	\$1.50
Powder, caps and fuse.....	0.85
Compressed air.....	0.15
Miscellaneous.....	0.90
Total.....	\$3.40

All development work was contracted at the following rates for breaking and mucking, not including timbering, the company furnishing the supplies:

Motor drifts (7 by 7.5 ft. in clear)

\$3.50 to \$4 per foot (depending on whether dry or wet ground)

Tramming drifts (5.5 by 6.5 ft. in clear)

\$2.25 to \$2.50 per foot (depending on length of tram)

Raises (4 by 5 ft. in clear)

\$1.20 to \$1.80 per foot (depending on length of raise)

At these rates, the contractors were able to make from \$3.75 to \$5 per shift, depending on the ground and their own efforts.

General Distribution of Costs

	Per Dry Ton
Labor.....	\$0.329
Powder, caps and fuse.....	0.091
Timber.....	0.049
Motor haulage.....	0.025
Gravity tramway.....	0.020
Supervision and engineering.....	0.038
Compressed air.....	0.024
Miscellaneous.....	0.111
Total.....	\$0.687

These figures include every item of expense incident to the mining of the ore, including the necessary development work. Under the head of miscellaneous is included the proper proportion of all general expenses, such as taxes, insurance, Salt Lake and New York office expenses.

HAMMER AND PISTON DRILLS USED

For all large drifts, such as motor drifts and stope drifts, and for widening out stopes, No. 7 or No. 9 Leyner water drills were used, the smaller machines being used only in soft ground. For small drifts the one-man 2.5-in. piston machine, of both Ingersoll and Sullivan makes, was used, while hammer drills were used for raising and stoping.

WATER SUPPLY

In the principal drifts and crosscuts on each level a 1½-in. pipe line was laid to supply water for the Leyner machines and for fire protection. A reserve supply of water was stored in tanks erected in a drift 50 ft. above the top level, the head being sufficient to carry the water to all parts of the workings. As the two upper tunnels were dry, it was neces-

sary to fill the storage tanks by pumping from the ditch on the main-haulage level, a small compressed-air pump operated an hour or two each day being used for the purpose.

DATA ON MOTOR HAULAGE

The main-level track is laid with 60-lb. rails, 36-in. gage, and a $\frac{1}{4}$ per cent. grade in favor of the loaded cars. Two General Electric 10-ton locomotives, each pulling a train of seven cars or 60 tons of ore per trip, have handled as high as 4,000 tons in a shift of 8 hr., the average being 2,500 tons. Loading from a chute in which the ore runs freely, a train could make a round trip in 15 min., divided as follows:

	Minutes
Loading train at ore chute.....	2.5
Trip to ore bin (4,000 ft.).....	5.0
Dumping train at ore bin.....	2.5
Return trip to chute.....	5.0
Total.....	15.0

GRAVITY TRAM FROM MINE PORTAL TO RAILROAD BINS

As the portal of the motor-haulage tunnel is on a hill side, at a vertical distance of 350 ft. above the level of the railroad yard, and at a horizontal distance of 700 ft., a surface gravity tramway is utilized to transport the ore to the yard. Two Stein trams were installed and were so arranged that they can be operated singly or both at the same time.

The surface incline has 60-lb. rails and a 36-in. gage, with an inclination varying from 30° at the top to 18° near the lower end. At the level of the motor-haulage tunnel a 300-ton ore bin gives sufficient storage to assure no delay to the motor trains. The gravity-tram skips are loaded from this bin through air-lift gates.

At the bottom of the incline a circular steel ore bin of 2,000 tons capacity gives sufficient storage to avoid delay in the operation of the tram even if railroad cars are not furnished for several hours at a time.

The skips on the gravity tram have a capacity of 12 tons, and dump direct into the steel ore bin through doors in the bottom of the skips. Railroad cars are loaded from the steel ore bin by means of air-lift gates, only 4 min. being necessary to load a 70-ton car.

One engineer at the head-house of the gravity tram can easily lower 4,000 tons of ore over this tram in a shift of 9 hr.

The brakes on the drums of these trams are tightened by dead weights, and released when the weights are raised by compressed air. When the supply of compressed air fails, the brakes are automatically applied, eliminating any danger of a runaway.

Cost of Operating Gravity Tramways

		Cost per Month
At capacity of 3,000 tons in 9 hr.:		
Labor:		
Tram engineer.....	\$135.00	
Loaders at bins.....	525.00	\$660.00
Supplies:		
Compressed air.....	\$13.50	
Miscellaneous.....	14.00	27.50
Repairs and Renewals:		
Labor:		
Repair man.....	\$82.50	
Miscellaneous.....	12.00	94.50
Supplies.....		60.00
Total.....		\$842.00
Cost per ton.....		\$0.0094

SAFETY OF MINING SYSTEM

As many engineers have gained the false impression that a system of mining involving the use of shrinkage stopes is necessarily hazardous, a list of the fatal accidents during the 3 years under discussion may be of interest. The following is a complete list of fatalities for a 3-year period during which time more than 3,000,000 tons were mined at a cost for timbering of less than 5c. per ton:

	Number of Fatalities
In stopes:	
Falling rock	3
Other workings:	
Falling rock.....	1
Man fell into ore chute.....	1
Electrocuted	1
Asphyxiated.....	1
Total.....	7

Most of these accidents were entirely due to the carelessness of the men injured. A noteworthy feature is that not a man was fatally or seriously injured through the use or handling of explosives.

CONCLUSIONS

The system of stoping as practiced was eminently satisfactory to supplement the steam-shovel operations without injuring the ore reserves of the property through a mixing of capping with ore. It had the

advantage that all ore produced was absolutely free from waste, since both stopes and development drifts were discontinued when capping was reached. The assay value of the ore produced could be regulated, and the tonnage materially increased or decreased without affecting to any extent the cost per ton.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Broken Hill Underground Mining Methods

BY E. J. HORWOOD, BROKEN HILL, N. S. W., AUSTRALIA.

(New York Meeting, February, 1916)

The varying physical character and large extent of the Broken Hill lode necessarily involve the employment of a variety of underground methods. The lode had its origin in an extensive fault plane traversing metamorphosed schists conformably, as a rule, with their beds of stratification. The underground waters carrying minerals in solution deposited their contents in the original cavities formed by the faulting action, and in the enlargements of these cavities due to dynamic forces brought to bear on the rocks, more especially on the hanging-wall side of the fault. This deposition was supplemented by metasomatic replacement of a portion of the original rock contents by the argentiferous sulphides of lead and zinc which form the staple products of the district.

Although the orebody is practically continuous throughout the mines, its width varies greatly, ranging from a few feet to about 350 ft. The widest portions occur in conjunction with huge folds in the inclosing country rock, almost exclusively on the hanging-wall side. The ore in these folds pitches to the south in the southern half of the field, and to the north in the northern half; there are, however, undulations in these ore channels evidently due to compression of the rocks in the direction of the channels. In the earlier days, before the orebodies had been opened up, vertical cross-sections across these bulges in the hanging wall gave the lode the appearance of the "saddle formation," so well exemplified at Bendigo in Victoria; but subsequent development of the ore channels has long since proved that the formations in the two districts are entirely different.

The depth reached by the zone of complete oxidation of the sulphides varies from about 250 to 550 ft. from the original outcrop, while partial oxidation extends in places below the 1,000-ft. level. The result of oxidizing influences has been the production of ore of every grade of cohesiveness, from that of dry sand to that of hard compact rock.

The methods of mining followed in this field, and even in individual mines, are varied in accordance with the character of the ore. The general practice of the field is to sink vertical shafts, generally on the foot-

wall side, free from the liability of disturbance from settlement due to stoping.

Where there is an assurance of depth of orebody, the levels are spaced at distances up to 200 ft., which is considered the maximum for economical working, for, although the cost of opening up each level per ton of ore commanded would be decreased by a further increase in the spacing, the extra costs due to extra wear and tear of chutes, reduced accessibility of the stopes, etc., would more than counterbalance that saving.

Shaft Sinking

The largest shafts in the district measure 13 ft. 8 in. by 9 ft. 6 in. within timbers, and are divided into three compartments—two for winding purposes and one for ladderway, and to accommodate air main, pump column, electric light and power cables, etc. The details of the shaft timbering are shown in Fig. 1. Each winding compartment carries cages capable of holding two ore trucks, end on, each about 25 cwt. capacity; draft horses are also sent up and down in these cages at the end of each shift.

Bearers, the ends of which are let into hitches cut in the solid rock, are placed about 50 ft. apart vertically, and below them, in addition to wood blocking, each wall plate is hung from that above by means of wrought-iron hangers, as illustrated, until the weight can be taken by the succeeding bearers. Strong frames are usually hung below the lowest wall plate to protect it from flying rocks, the result of blasting operations when sinking.

The maximum distance of the timbers from the shaft bottom during sinking varies according to the nature of the ground. The general practice in sinking these shafts is to employ four reciprocating rock drills on two stretcher bars, and to arrange the holes so as to be able first to fire out a central cut across the shaft, after which the holes adjoining the cut are fired in succession by using varying lengths of fuses. A depth of about 6 ft. is gained with each firing. Electric firing has been tried on various occasions, but was not adopted because of the greater economy in explosives resulting from the use of time fuses, which enable the burden on the various holes to be successively reduced.

In sinking Delprat shaft on the Proprietary mine from the surface, the bottom of the shaft was illuminated by means of rays of electric light thrown down the center by means of a parabolic reflector and a mirror placed on the surface at an angle of 45°. In order to avoid shadows, the dividing timbers were left out until the sinking was completed. Operations were thus materially facilitated, especially as there was a considerable quantity of dripping water. Powerful first-motion engines and buckets of 20 cu. ft. capacity were used for sinking direct

from the surface, but in deepening existing shafts considerations of available room tend to restrict the size of the hoists, which, in such cases, are usually geared. As these lifts, however, are smaller, rapid hoisting is not so important. Travelers, whose depth, to prevent jamming, is not less than twice their width, are used to guide the bucket through the timbering; these work in ordinary runners.

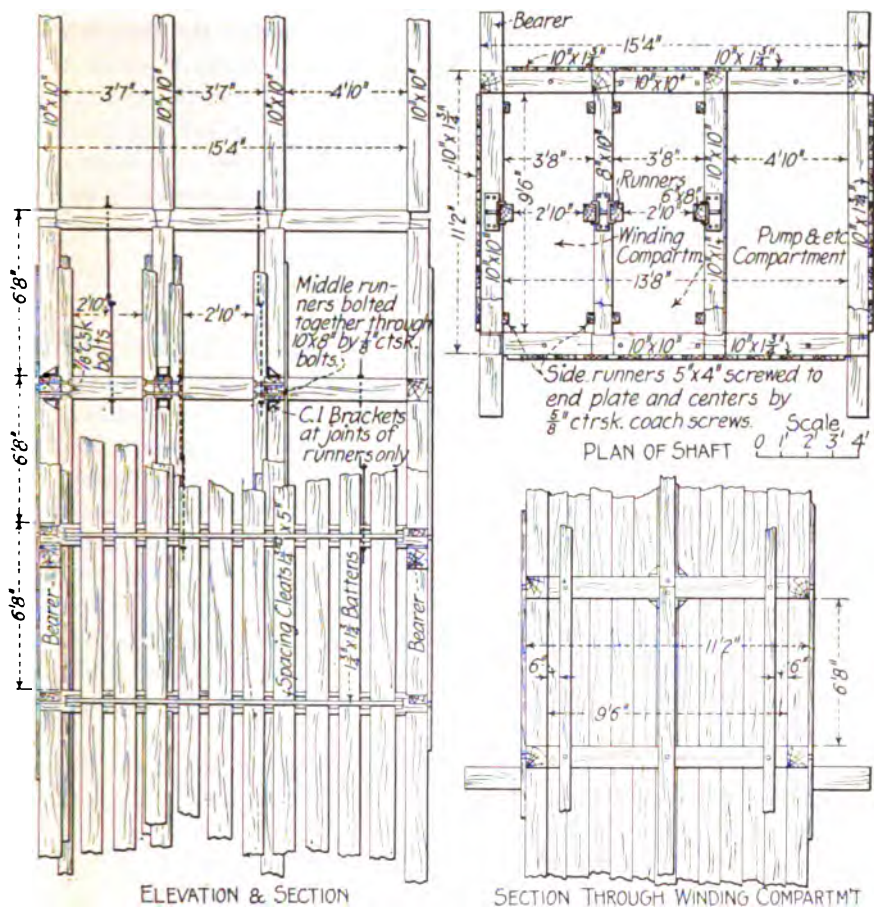


FIG. 1.—PLAN OF DELPRAT SHAFT WITH DETAILS OF TIMBERS, BROKEN HILL PROPRIETARY CO., LTD.

The Kinetore shaft on the Central mine was recently sunk for a time with the use of hammer drills held in the hand, the maximum depth of hole being from 1.5 to 2 ft. The system was, however, not persisted in, because with the particular type of drill used the effect of the vibration was too severe on the men using the machines continuously for the considerable periods necessary, and, in addition, the substitution of short lifts for the deeper sinks to which they were accustomed was not favored

by the contract miners. So the reciprocating drills were finally reverted to. There seems, however, to be every probability that shaft sinking may be advantageously carried out by this method with machines having less vibration, and further attempts will no doubt be made in that direction.

Where new shafts are being provided to command existing workings, the practice is to put up rises about 6 ft. square from the various available levels, subsequently stripping the sides to accommodate the shaft timbering. Rising is effected with the modern air-fed hammer drills using water, either through hollow steel or by means of sprays, to allay the dust. By this method shovelling is avoided; the expenditure in explosives is reduced; delays caused by baling water are eliminated; and power is saved in hoisting, since the broken rock can generally be used for stope filling on the various levels.

Winding Engines and Equipment

The general practice of the field is to employ first-motion, double-drum steam engines, fitted with auxiliary steam cylinders controlling the brake release and reversing gear, and thus reducing to a minimum the physical exertion of the drivers. The function of the auxiliary controlling the brakes is to keep the brakes off, as counter-weights supply the power for the brakes, which would be automatically applied in the event of failure of the steam appliances. Post or pillar brakes are generally used with Ferodo linings, those having parallel motion giving the best results. Piston valves are generally used on these engines, the cutoff being generally at a fixed position and full steam applied until the cage is within a safe distance of the surface, when the steam is turned off; but in the case of the South mine more elaborate expansion gear is employed with satisfactory results.

The exhaust steam from the principal winding engines of the district is discharged into accumulators feeding the exhaust-steam turbines which furnish electric power for general purposes.

Revolving indicators enable the driver to watch the position of the cages, and in the case of Delprat shaft on the Proprietary mine the view of the brace, etc., is obstructed so that the driver's attention shall not be diverted, thus requiring him to work only according to his signals and indicators.

On the larger engines plough-steel wire ropes of Lang or Albert lay are used, $1\frac{1}{4}$ in. in diameter with a factor of safety of about 10 or 12 on dead pull when new, and having six strands of 17 or 19 wires, the former where the pulleys and drums are of ample diameter.

On the Proprietary mine springs are used, as shown in Fig. 2, near the attachment of the ropes to the cages to take up the shock when the

weight is coming on to the rope with a view to increasing the useful life of the ropes.

Regarding safety appliances to guard against overwinding and breakage of ropes, in addition to those on the cages (on which grips engaging the sides of the timber runners are favored), safety hooks of the well-known Ormerod type (see Fig. 3) are used, which disengage the rope on passing through thimbles placed below the pit-head pulleys and support

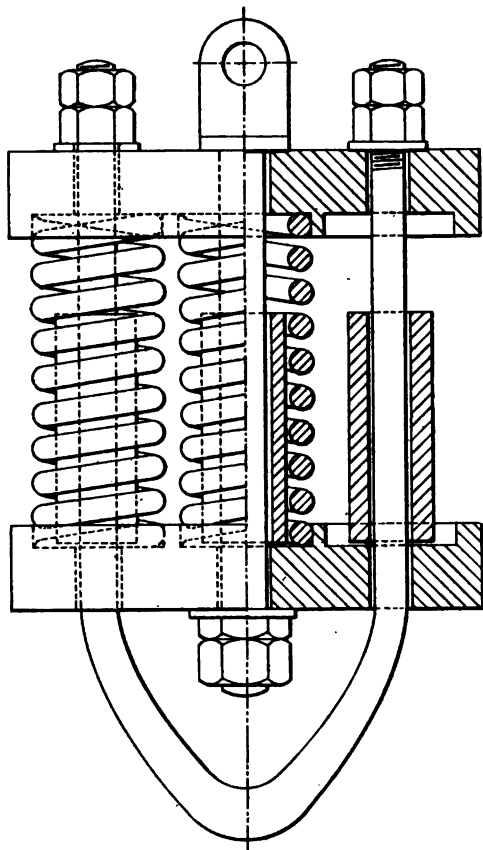


FIG. 2.—SPRINGS FOR REDUCING SHOCKS TO WINDING ROPES.

the cage. In addition, safety chairs are fixed above the top brace to catch the bottom of the cage if the other appliances fail.

At all shaft entrances lattice-work iron doors, balanced, are provided to prevent persons entering the shaft, and in addition heavy bars of steel rail, also balanced, are used to prevent trucks entering the shaft.

Poppet heads of the gallows-frame type are in general use, constructed of steel lattice framing, built clear of the engine house, which is generally of masonry.

The equipment of the top brace, or landing, includes a revolving tippler on to which a truck of ore can be tipped by an ore inspector whose duty is to bring to the notice of the underground manager any cases found where mullock in undue quantity, or pieces of ore too large for the breakers, are included, in which cases offenders are punished by suspension or dismissal, according to circumstances.

As a prevention against fire being communicated from the surface

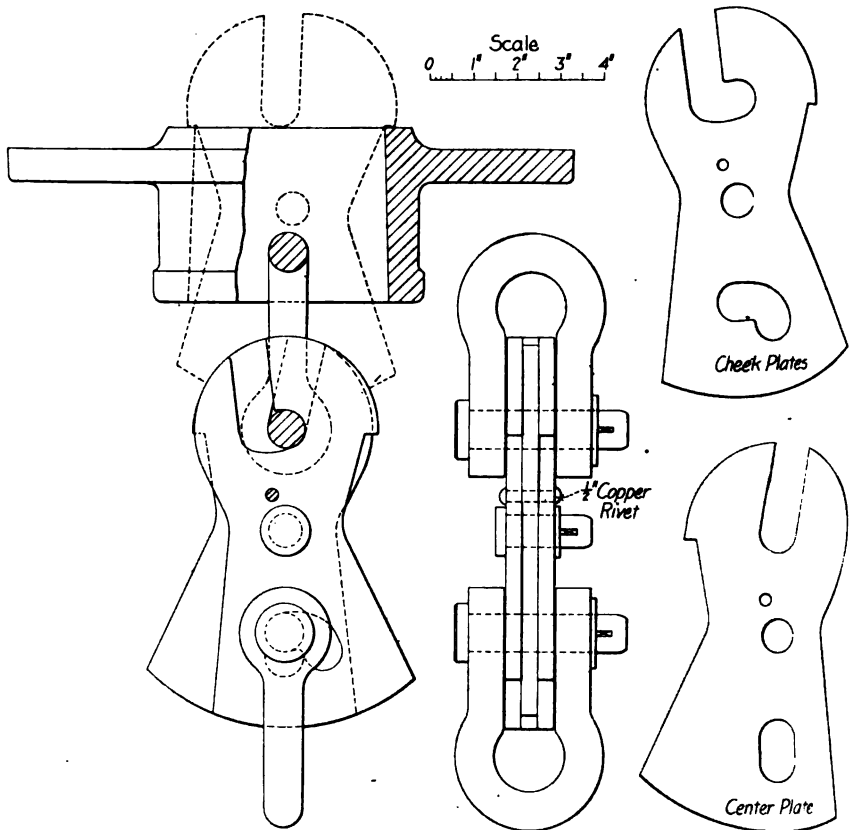


FIG. 3.—ORMEROD TYPE OF SAFETY DETACHING LINKS.

works underground, or *vice versa*, a section of the shaft timbering on the Proprietary mine is cut out just below the surface chairs; heavy plate-iron doors are hinged to the sides and can be closed at short notice. Like all the safety appliances, these are thoroughly tested periodically to insure their efficiency when needed. In conjunction with these, plate-iron doors are provided at all approaches to the shafts to control the air currents in case of fire.

In only one instance in the district, namely, at the Zinc Corporation,

is the system of skips with ore pockets below the plats used. It is found generally more convenient, in view of the moderate tonnages hoisted at individual main shafts, to use cages, which are then always available for use in hoisting and lowering men, horses, and material as well as the ore trucks. Moreover, the latter system enables the weight

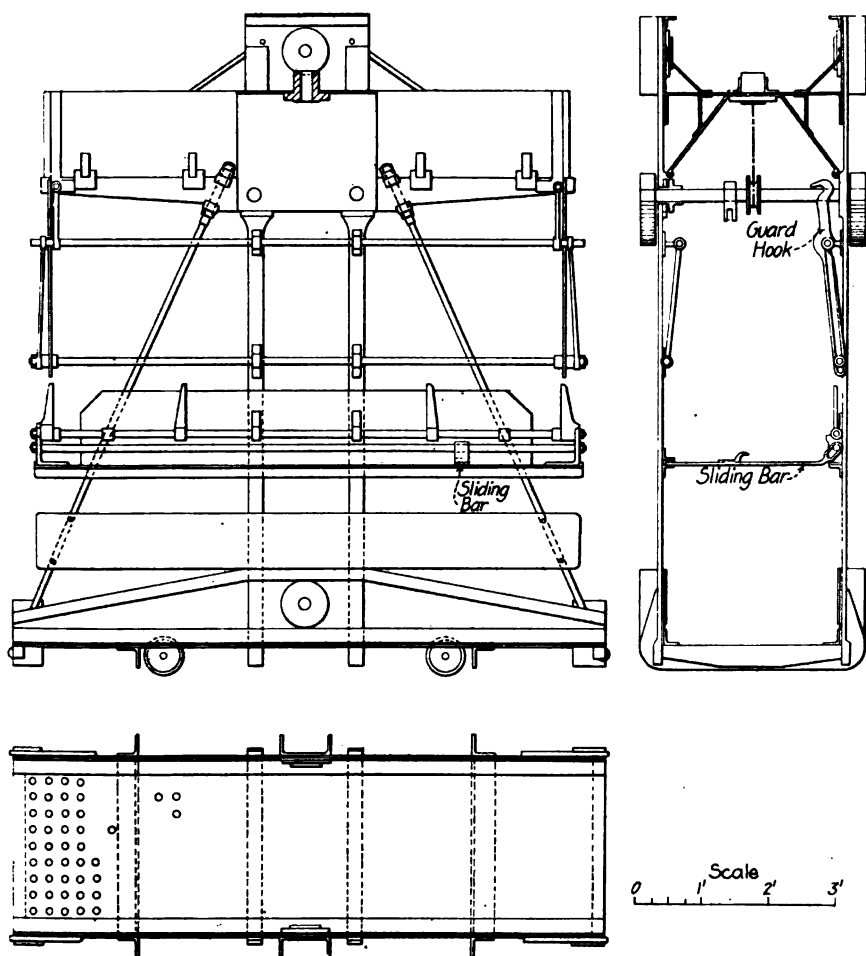


FIG. 4.—CAGE ACCOMMODATING TWO ORE TRUCKS.

of each party's trucks of ore from any number of levels to be definitely ascertained at the single weigh-bridge on the surface.

In the case of the Zinc Corporation, owing to the regularity of the orebody and (in the upper levels) its moderate width, it was practicable to pay contractors by measurement, so that the disadvantage last mentioned as attaching to the use of ore skips does not exist.

At Delprat shaft on the Proprietary mine, where as many as 1,650

trucks of ore have been hoisted from about six levels in an 8-hr. shift, after changing half the number of men employed on the shift and with the usual 20 min. for crib, a back-shunt is provided at the back of each plat, into which the empty trucks are pushed; from the end of this back-shunt the trucks automatically gravitate to a line parallel with the full lines and laid on the requisite grade, and of sufficient length to accommodate 25 or 30 empties, which are taken away by horse traction in "rakes" of 10.

The use of cages (Fig. 4) capable of carrying two trucks, end on, facilitates rapid caging, and this is further assisted by the use of a sliding bar which retains the empty trucks in the cage on the loading side of the shaft, and *vice versa* as regards the loaded trucks, rendering it unnecessary to handle more than one bridle at each caging or uncaging of trucks, as the sliding bar is automatically pushed to and fro into position by the moving trucks.

For horse traction, the general practice is to use trucks with wheels fast on the axles; but where the distance of the shaft from the ore chutes is short, hand-trucking is practiced, and in some of these instances one wheel only on each axle is fast, facilitating trucking on curves.

At the South mine a system of automatic signals has been introduced for the purpose of indicating to the engine-driver whether the chairs are in or out for each level; this is achieved by the use of electric contacts attached to the chairs and a series of electric lights in front of the driver in the engine room. Although this device is claimed to have proved itself reliable, the managers of other mines, where the shafts are probably wetter and short-circuiting is, therefore, more likely to occur, have preferred to guard against chairs being left in by requiring the driver, in changing over to a new level, to lower the cage slowly when passing levels for the first time.

Pumping Facilities at Main Shafts

As the various levels are laid out on a gradient of about 1 in 190, or steeper, toward the shaft, the drainage of the mines is conveniently concentrated at the shaft plats, where excavated tanks are provided below the rail level at the pumping stations to hold sufficient water as sumps. Electrically driven geared pumps of various makes are used, pumping against heads of from 400 to 1,000 ft.

Communication between Surface and Underground

Telephonic communication between the surface and the various levels and isolated portions of the principal mines is established by connection of these places with the surface mine exchange. Hoisting signals are given by means of balanced wire ropes, the moderate depths of the local mines permitting the satisfactory use of this means of communication.

Ventilation

Artificial ventilation is generally practiced on the field, fans of the Capel type and of about 70,000 cu. ft. per min. capacity against 3-in. water gage being most common. Other types of similar capacity represented here are the Waddell and the Sirocco. On the Proprietary mine, which covers $\frac{3}{4}$ mile, are both a Capel and a Waddell fan. In general

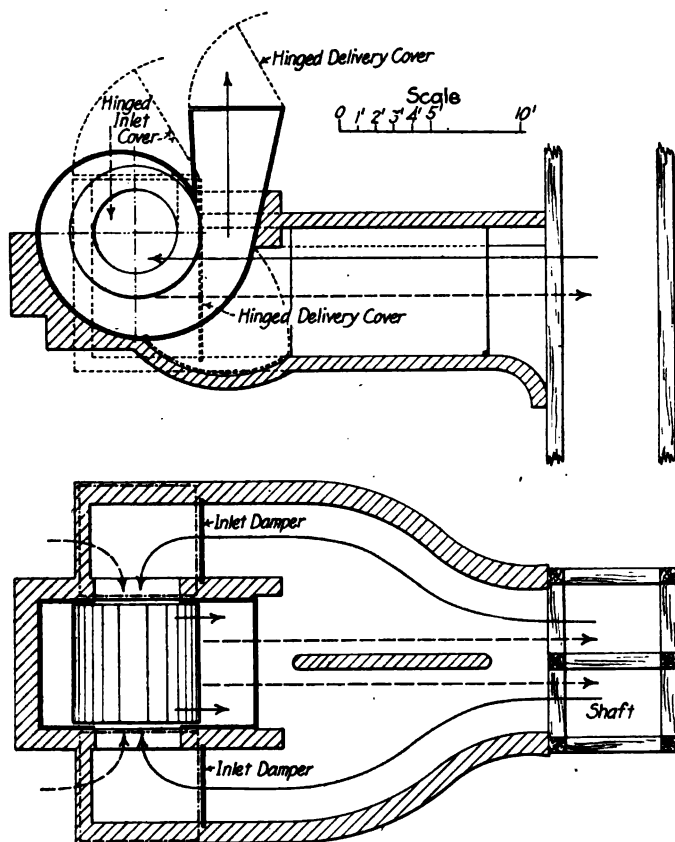


FIG. 5.—ARRANGEMENT OF AIR DUCTS, DOORS, ETC., FOR REVERSING VENTILATING CURRENTS.

use these fans exhaust the air but ducts and doors are provided, as shown in Fig. 5 by which these fans can be promptly made to act as pressure fans in case of fire making a change of draft advisable.

General

Magazines capable of holding sufficient explosives for 24 hr. are established on each important level, and generally, in conjunction with

them, stores of tools are provided, the rule being that no new article will be issued before the return of the old. All tools, except drills and picks, are branded with numbers for identification. Explosives are issued to contractors and charged against their contract, at the expiration of which a statement of earnings and all details is furnished to them. The majority of the contracts are for 4 weeks; but in some cases where the working conditions are liable to change considerably during the period of contract, they are for 2 weeks only. In some other cases, where conditions remain unchanged, contracts are sometimes let for several months.

Tools are provided free of charge, but any loss sustained other than through fair wear and tear is chargeable to the contractors, who constitute the great bulk of the workmen. Wages-work was general in stopes up to 1892, but the efficiency of the work fell so low that contract, or piece-work was insisted upon, and generally introduced after a strike lasting 18 weeks. Owing to the higher earnings prevailing under this system, the majority of the men now favor contract work, which, in view of the inherent difficulties of supervising wages-men in so many isolated localities, is the only practicable means of securing efficiency.

Underground Development and Extraction Levels

The general practice of the field, except where the lode is narrow, is to place the extraction levels in settled country, at a safe distance from

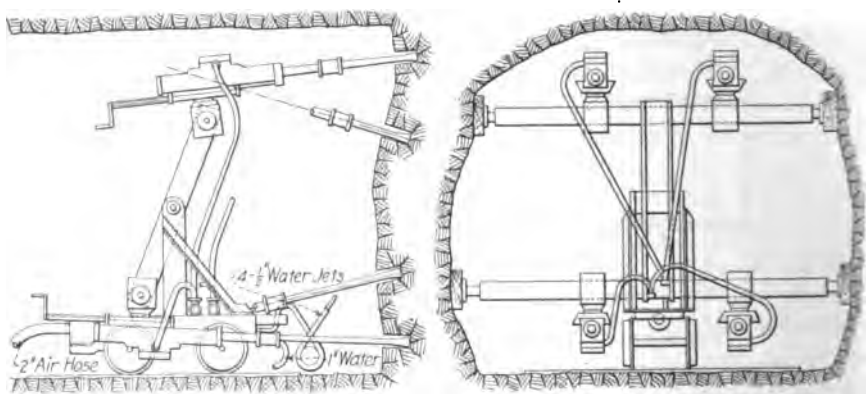


FIG. 6.—ROCK-DRILL CARRIAGE USED AT CENTRAL MINE.

the lode on the foot-wall side. These are generally about 8 ft. wide, and carry double tracks.

Where the orebody is wide, crosscuts are run through the lode every 100 ft. In some mines where time of completion is of great importance, a special rock-drill carriage run on rails is used, on which four drills are mounted, enabling the face to be bored out quickly. The

best of these, namely, the one used on the Central mine, is illustrated in Fig. 6. The most economical method of driving, where time permits, is to allot two faces (if available) at convenient distance apart, to each party, so as to permit the handling of the broken rock by truckers, thus limiting the miners to operations requiring more skill.

The favored system of firing is that in which the whole face is bored out before dismantling the machines, the holes for the initial cut (which is generally formed in the center of the face) being fired first, of course, and the surrounding holes in rotation, so that by varying the lengths of fuses the burden on each hole may be lightened by the preceding firing (see Fig. 7).

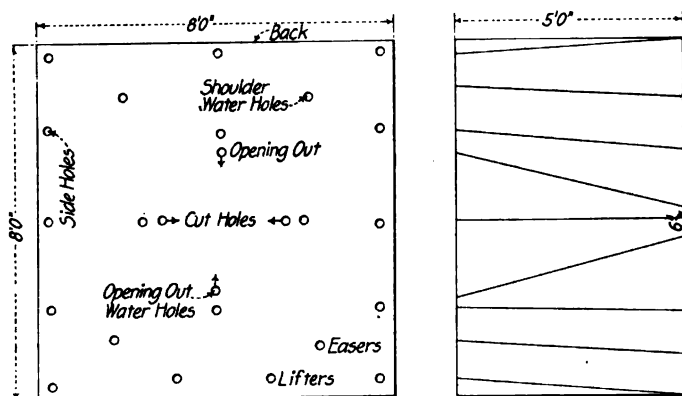


FIG. 7.—ARRANGEMENT OF HOLES FOR BLASTING OUT DRIFT.

Rises

Since the advent of the air-feed or telescopic hammer drill, vertical connections have been made to a greater extent than formerly by this means instead of by winzes. The advantages of rising are numerous. There is less risk of injury to miners, as the danger of stones or tools, etc., falling from bucket or sides of winzes is absent. The facilities for handling the broken rock are greater, shovelling on a rough bottom being eliminated. The engine-driver and bracman are dispensed with. There is improved ventilation, since the timbering of the rise can be carried closer to the face than in winzes, enabling effective use to be made of incoming fresh air whether induced by fans, air or water jets in ventilating pipes, or by curtains in adjoining drifts. The light weight of modern rising drills facilitates the work of rigging the machines in the most advantageous positions. Finally, in rises there is the absence of trouble from water; the greater facilities for detecting missed holes; and the fact that missed holes are less likely to occur in rises than in winzes, which are usually wet. Another advantage of rises as compared with

winzes is the better ventilation of the former, heavy CO₂ gas tending to lie on the bottom of the winze.

The modern hammer drill makes much less dust than the reciprocating drill, and the small amount produced is easily laid by using water in the hole through hollow steel or water sprays playing on the collar of hole. Although for the past few years the Regulations under the Mining Act require a water supply wherever rock-boring machines are in use, it may be mentioned that for years before this rule came into force, all the hard-ore stopes in the Proprietary mine were reticulated with water under good pressure, enabling water jets to be used when boring; it was recognized that any condition tending to injure the health of the workmen called for remedial measures, not only for humanitarian reasons but in the interests of the industry generally.

It has sometimes been sought to limit the height of rises by statute, but this is unnecessary, since excessive heights are prohibitive by reason of expense in handling tools, timber, etc. Various systems of timbering are employed. In some cases the rise is simply divided into two compartments by timber, lined on the side which is to be used as a chute for the broken rock. The other compartment carries the ladderway, water and compressed-air pipes, and 10-in. pipe for ventilation. Two square sets are also used, especially where the ground is not hard, and the extra size of the rise is therefore not objectionable on the score of expense.

Another method of timbering is the "box" rise, in which three compartments are formed in the same way as the two above described, the central compartment being used as a chute, and kept always nearly full.

By placing a curtain in the drift or crosscut opposite the chute, the air is made to travel up one side of the rise to the top and down the other. The extra length of this class of rise leads to increased cost, but those favoring it claim an advantage through having no ventilating air pipes to trouble about and to protect when firing—a precaution often neglected by the men.

Trucks

A simple form of ore truck is used where horse traction is employed. The bearings are so made as to permit the automatic greasing of the axles in passing revolving greasers. End- and side-tipping trucks are used for handling mullock, etc., and for filling stopes.

Methods of Stopping

Where the ore is friable, and therefore not self-supporting, the square-set system is now universally adopted, using sets of sawn 10 by 10-in.

Oregon pine, generally 6 ft. square and 8 ft. high from center to center. Because of several serious underground fires, attempts have been made, especially on the Proprietary mine to replace this system by the crosscut and others. In the former case, successive horizontal slices about 6 ft. wide by 7 ft. high were taken out and filled, and any drift timber used in the lower floor was drawn when the floor above was being stoped, tapered legs being used to facilitate this. Owing to the increased labor involved to both miners and mullockers, and the slow rate at which large bodies of ore can be worked, square sets were reverted to.

In places where the ore was sufficiently self-supporting, sloping stopes taking out slices 10 to 15 ft. wide were tried. The cost of handling the ore and mullock filling was very low; but by reason of variations in the strength of the ground what appeared a perfectly safe width at one time might be unsafe later on; and this system was also discarded, except in isolated cases.

In using square sets for stoping these large masses of ore, it was the rule in the earlier days to work out the ore in stopes 100 ft. high. Although this has been successfully done in many cases, it is now held by those who have had the heaviest ground to work that in the long run it is much safer and more economical to divide the height into two lifts of 50 ft. each. When stopes as high as 100 ft. are worked, the accumulated shrinkage due to the drag of the filling on the timbering in the stoped-out ground is very apt to leave considerable cavities between the old bottoms overhead and the top of the stope, rendering the blocking at the top of the stope useless. In some cases the old bottoms and filling above gradually subside and harmlessly follow the top of the stope below, but in too many cases there is a sudden drop of hundreds of tons on the top of the working stope, the effect of which may be its complete collapse, and great expense in making fresh arrangements to take out the ore, much of which will have fallen into and mixed with the filling, besides being much more expensive to mine than if the collapse had not taken place.

Where the ore is sufficiently cohesive to enable driving laths to be dispensed with to support the back, overhead stoping is practiced, because the ground can be blasted to greater advantage, and with less explosives per ton, than in underhand stoping. In heavy ground, however, or, as a rule, where a slice of ore under old stope bottoms is being taken out, underhand stoping is carried out, the miners first securing the ground immediately above the set or sets to be carried down, this securing being done with the aid of cantilever booms supported on posts set on the cap at the bottom of the set, driving laths being used if the condition of the bottom requires them (see Fig. 8).

The back or backs of the sets, if more than one are being carried down at a time, having been secured, the ore is then removed for the

full dimension required to accommodate the square sets, horizontal breasters of 10 by 2-in. timber with joggled ends being inserted to secure the sides; and when these reach the floor of the set, the square sets themselves are placed in position and blocked. Before taking out the next lower set, the bottom timbers of the square set, in the case of the Proprietary and some other mines, are hung from the top timbers by nailing two laths to each pair of timbers, or, as in the Central mine, by connecting adjoining timbers by means of iron rods about 1 ft. long having spiked ends, about 4 in. long, bent at right angles, that can be driven into the timber. The broken ore is put into chutes formed in the square sets in the usual way, and moved as the stoping advances.

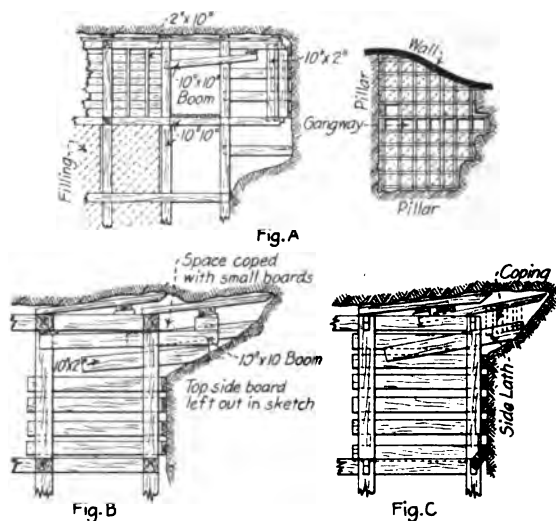


FIG. 8.—(A) METHOD OF STRENGTHENING PADDOCKING LATHS AND DETAILS OF UNDERHAND STOPING. (B) DETAILS OF DRIVING LATHS, BOOM ETC., SUPPORTING BACK IN RUNNING GROUND. (C) DETAILS OF DRIVING LATHS, BOOM ETC., SUPPORTING BACK IN RUNNING GROUND.

When all square sets from top to bottom of the run have been put in, the hanging laths and irons are, of course, removed and used in other sets as required. There is very little strain on these hangers as all the top weight is taken by the boom in the top set and the weight of the sets below is taken up by the blocking.

In stoping ore of a dry sandy nature, common in the oxidized zone of the Proprietary mine, great assistance is obtained from the use of water, sufficient of which is used in a narrow trench to soak into and wet the ground against which the breasters will be placed. By this means the danger of a run of ground is reduced to a minimum.

Where back-laths have to be driven in dry running ground, the work is

greatly facilitated, besides being made healthier, by wetting the material in the back with the aid of pointed pipes having outlets on the sides.

Another practice in vogue in the Proprietary and other mines is the use of screw jacks for driving the laths "home." Hammering not only destroys the lath before it gets home, if driving is difficult, but the vibration tends to set up a run of ground.

With regard to the location and size of square-set stopes, the practice generally adopted is to carry the stope across the full width of the orebody, so that the stoping advances in the direction of the course of the lode, thus minimizing the effects of pressure on the stoping from the hanging wall, which, of course, becomes heavier as stoping proceeds. The extent to which stoping is allowed to advance before being filled depends on the prevailing conditions; but, in general, two sets wide by as many sets across the stope as possible are filled at each mullocking, leaving one vacant run of sets against the face. In the case of heavier ground, it is often necessary to paddock off the sets close to the working face, leaving single vertical gangways for the full height of face to give the required points of access to the face for re-starting the stoping.

The material most commonly used for filling is the residues from the flotation plants, but in the case of the Central mine, where there is much heavy moving ground and where mullock filling is obtained cheaply from a quarry immediately over the workings, the use of tailings was discontinued. Where tailings are used, as in the Proprietary mine, the sets are closely lined with 10 by 1-in. Oregon planks, and these are strengthened by two upright 10 by 2-in. laths wedged tightly between the caps. Where mullock filling is employed, 5 by 1.5 to 2-in. laths spaced 4 in. apart are used, buttressed where necessary in the same manner as is done with the 1-in. planking.

These buttress pieces are removed just prior to the filling of the succeeding stope. The bottoms of the square sets are covered with 2-in. planking, the space between which and the solid ore is filled with ore, to prevent breakage of the planking under the pressure of the overlying filling.

Open Stoping with Bulkheads

Where the sulphide ore is still in its original solid condition, free from decomposed veins and vugs, stoping is carried out without the use of square sets except for chutes and ladder-ways. The stoping begins at the various crosscuts, and where these are not already connected by driving along the orebody, a stope about 20 ft. wide by 16 ft. high is carried each way, generally along the foot wall, with a view to connecting with the adjoining crosscut and establishing ventilation. The sill-floor stope is gradually extended until the entire width of the lode is stoped out for the height above mentioned. These excavations frequently

exceed 100 ft. in width, and may in some cases attain a length of 300 ft. Pigsties, or bulkheads, 5 or 6 ft. square, of 10 by 10-in. timbers crossed, are built from 12 to 15 ft. apart to support any ore likely to flake off. Excavations of this size are not common, as it generally happens that pillars of ground, either poor ore or country rock, occur that serve to support the back. No hard and fast rule can be laid down as to the size of these excavations, so much depending on the nature of the ore, its freedom from floors or horizontal cracks or seams, the nature and configuration of the inclosing country rock and walls, which may converge as depth increases, or *vice versa*, making wide stopes more or less safe respectively.

The extent to which pillars are left, whether because too poor to mine or because needed temporarily as a support to a shaft or other working, is also a factor affecting the area which can safely be undercut. Each case must therefore be decided by the management in view of all the conditions.

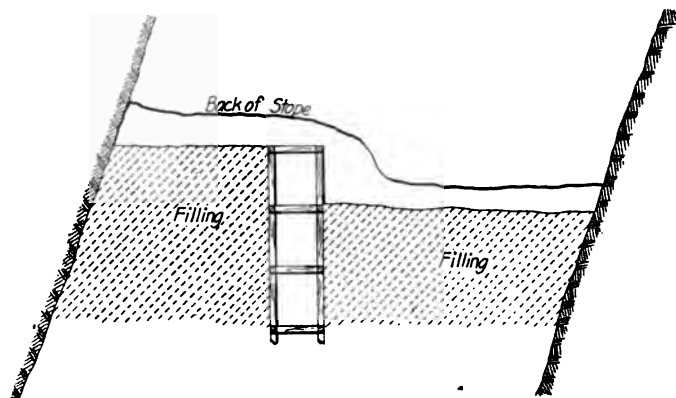


FIG. 9.—OPEN STOPE IN TWO FLOORS WHERE HANGING WALL IS WEAK.

The work of placing in position the permanent main drift and cross-cut timbers can proceed as soon as blasting operations on the side of the stope are sufficiently remote to permit this to be done without risk of the timbers being thrown down, and when these have been placed and a sufficient area of filling has been put into the stope, the second stope can be started, allowing the rest of the sill-floor stoping and filling to proceed simultaneously with the stoping of the second slice. This and each subsequent horizontal slice is taken out 8 ft. high. The general practice of the field is to cover the sill floor of the stope, *i.e.*, on the level, with 4-in. planking to enable the stoping below to approach the old bottoms with safety; possibly 3-in. planking might be substituted to advantage.

The filling is taken out of chutes into end- and side-tipping trucks, and tipped over the whole area of the stope, and the space between the top of filling, *i.e.*, the rail level and the back of stope, usually from 4 to 5 ft.

high, is again dotted with pigsties to support the back. The bulkheads are usually removed as the filling advances, although occasionally one is left and buried should the condition of the back render this necessary. It is considered safer to build bulks at regular intervals rather than only at such places as the supervising officers consider necessary, but to increase the number if occasion requires it.

In stoping hard ore against a weak and overhanging wall in the Proprietary mine, the open stope was worked in two floors (Fig. 9), the higher of which was against the wall in question, and by this means a minimum extent of wall was left unsupported, as filling was introduced to support the wall as early as possible, and the back, as seen in cross-section, was thus kept in the most favorable shape to resist and support the hanging wall.

The height to which these large stopes can safely retain their full

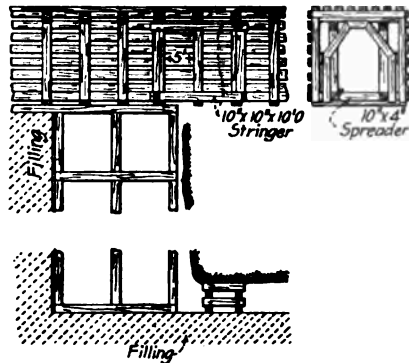


FIG. 10.—METHOD OF SUPPORTING DRIFT TIMBERS WHEN BEING UNDERMINED BY STOPING.

width depends on the conditions regulating their extent in the first instance, but the local practice is to discontinue using the open stope or bulkhead system when there remains from 25 to 30 ft. of ore under the level above. This slice is stoped out on the square-set system, underhand, if the ore is sufficiently cracked and broken to require it (Fig. 10). The bottom of the square sets is laid on the filling, and especially where tailings are used, scrap timber is first laid to give added support.

As the mining of these arches involves less heavy firing than in the open stopes, light reciprocating drills of various types, weighing about 120 lb. and capable of being quickly rigged on stretcher bar, are generally used. The ore in these cases is more or less broken as the result of the enormous pressure concentrated on the arch from the hanging wall, and in consequence does not lend itself to heavy firing, especially in contact with the square-set timbering which might be shot out, bringing about the loss of the stope. In some cases, the ore is sufficiently cracked

and broken to make the use of hammer drills held in the hand advantageous. Difficulty is met with in getting the miners always to use water with these drills in the interests of their own health, and this also applies to the ordinary "popping" of rocks for which these drills are also used.

For the short holes necessary, the drills will generally clear themselves with the aid of the water spraying into the hole, and the exhaust air from the machine; but in many cases rather than submit to being slightly splashed with mud, men risk their health by working the machines dry and inhaling dust unnecessarily. It is too often forgotten that when hand drilling was employed for popping purposes the mud had to be periodically scraped out, and in those rare cases in which the popper does not clear itself the hole can be scraped out as of old.

Reciprocating rock drills with cylinders $3\frac{1}{4}$ in. in diameter are generally used in the solid ore, since with these, approximately horizontal holes can most conveniently be bored in the backs, enabling regular and reliable roof to be formed. Telescopic drills were tried for general stoping, but these tend to leave an unsafe back.

As many holes as practicable are bored at each setting up, and these are fired by ordinary fuses cut to lengths to give the desired rotation of shots, blasting gelatine (92 per cent. nitroglycerine) being used so as to effect the maximum of shattering of the ore to save spalling. For the same reason, simultaneous firing with electric fuses was, after making tests, found uneconomical, as the ore came down in much larger pieces entailing added expense in "popping" and spalling. The firing of one hole at a time results in numerous cross-breaks in the ore and reduces the cost of breaking up the ore to the size required by the mills, i.e., about 10 in. in diameter.

Another disadvantage of electric firing in stopes was that the excessive vibration set up in firing all the explosives at one time was a source of danger, as it would tend to weaken the backs of stopes and lead to accidents through falls of ground.

A modification of the open-stope or bulkhead system of stoping was made use of on the Proprietary mine, where it was desired to obtain the maximum output of ore from a given length of lode; this applied particularly in exceptionally hard portions of the lode, where the cost of sinking winzes was excessive, and the lode had less than the usual width. The system may be described as an adaptation to lode mining of the long-wall principle in coal mining. Several floors of stoping are worked at the same time, access to each face from the main mullock chute and ladderway being preserved by the provision of timbered drifts connecting the mullock chute with each stope. In this manner, any number of faces can be worked from each main mullock chute, each on a separate floor, but the maximum number actually operated was four.

The system was not extended, since the introduction of the Calyx

drill enabled mullock passes suitable for sand filling to be provided at about one-quarter the cost of winzes. The holes bored by the Calyx drill were about 10 in. in diameter.

Rill Stopes

A certain amount of rill stoping has been practiced on the field where conditions favored that system of work, but this is not usual. The hanging wall of the lode is oftener weak than strong, and constitutes a menace to the workmen employed on the sloping surface of the filling, especially when near the bottom of the rill where the danger is greatest, owing to the length of time the wall has been exposed.

The nature of the filling generally available, namely, sandy tailings which never form a compact surface, would necessitate the covering of the stope with planking to prevent undue mixing of broken ore with the filling.

Further disadvantages as compared with flat back-stopess are the greater difficulty of rigging the rock drills, and less favorable conditions for blasting, in that the ore is inclined as a rule to lie in horizontal layers, and lastly, greater difficulty in breaking and keeping separate worthless material occurring in the lode.

Transportation of Filling

The handling of mullock filling from the main chutes to the various stopes is to a great extent accomplished by horse traction on the Proprietary and some other mines, but where the distances are small hand trucking is more convenient.

Conveyor belts are used in a number of cases, particularly in the North mine, the Proprietary, and the Junction North, where the quantity to be handled warrants the installation. In the case of the North mine, the regularity and compactness of the orebodies has enabled the management to install a complete system of conveyors for delivering the filling to the chutes commanding the various stopes. Where the orebodies are irregular and spread over considerable distances this system cannot, of course, be economically employed.

Electric traction was introduced on one of the principal levels of the Proprietary mine, but was replaced by horse traction after being thoroughly tested. The distances to be trucked were not sufficient to compensate for the extra cost of attendance. The horse-driver acts as a trucker, also, in getting his rakes together and requires no assistant, but the more highly paid motor-driver needs an assistant to shift points, couple and uncouple trucks, etc. In addition, repairs, when necessary, are costly.

Having arranged the gradients of the levels so that a horse can draw the same number of loaded and empty trucks with equal ease, *i.e.*,

about 1 in 190, it is found that one horse can thus handle 10 trucks at a time containing 25 cwt. each.

Main-Drive Timbering in the Orebody

Various types of timbering for the main drives are employed, all of which give satisfactory results. Where hardwood timber is obtainable at

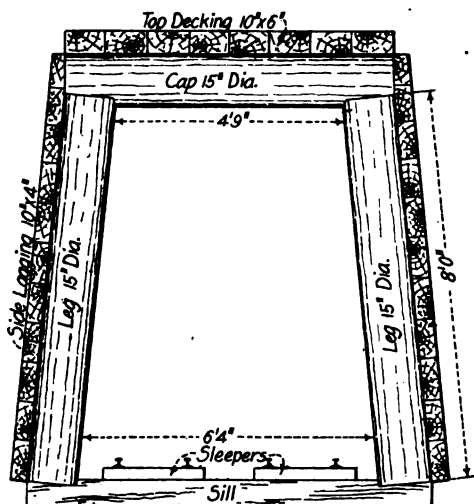


FIG. 13.—STANDARD DRIFT TIMBERS, SOUTH MINE.

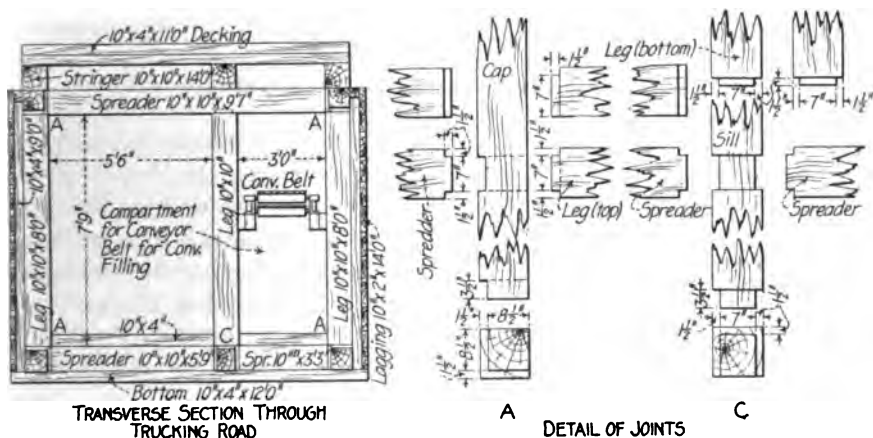


FIG. 14.—MAIN LEVEL TIMBERING, NORTH BROKEN HILL, LTD.

reasonable rates compared with Oregon, its use would be advantageous in these places. The principal methods of timbering main gangways in the lode are illustrated in Figs. 11, 12, 13, and 14.

A number of the mines use a considerable quantity of round hardwood

logs for ordinary drives in heavy ground, as their resistance to crushing is much greater than that of Oregon. On the Proprietary mine these are used in conjunction with old 80-lb. rail material as caps; this is specially advantageous where head-room is deficient.

Timbering of Ore Chutes

The majority of the ore chutes in the stopes, whether square-set or open stopes, are formed by two square sets lined with 10 by 4-in. hardwood planks, spiked to the caps in a vertical position. The cost of repairs is reduced materially by the use of hardwood, in spite of its extra price.

The use of square sets is advantageous in that repairs to the lining of chutes can be readily effected. On the British mine these chutes are built of 10 by 10-in. timber, having joggled ends, the idea being that the

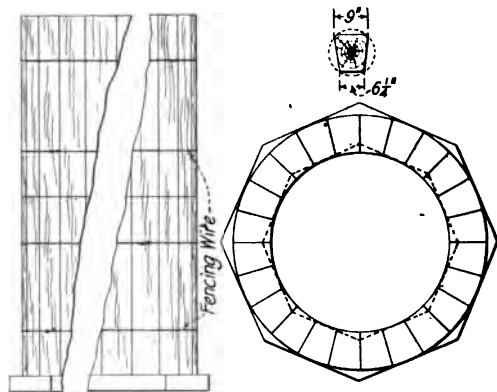


FIG. 15.—CIRCULAR ORE PASSES, SOUTH MINE.

timber will last out the stope. Against this, there is greater wear on the timber when the grain is at right angles to the falling ore.

With square sets a convenient construction is afforded for the provision of a ladderway alongside the chute, which, besides being useful for traveling, is of great assistance in case the ore in the chute "hitches up." The advocates of chutes without ladderways claim that the material can generally be started with the aid of a cannon, but this involves a certain amount of delay, especially as the cannon is not always effective.

On the South mine, where the ore occurs to a greater extent in pipes or chutes, circular ore chutes built up in sections, as shown in Fig. 15, are successfully used. The timbering is thickest, of course, near the bottom, where the wear is greatest. An essential condition for the satisfactory use of these circular chutes is that the pressure be approximately equal on all sides; their use, therefore, in ground subject to much pressure from the hanging wall, is prohibited.

Prevention of Underground Fires

The leading mines, where considerable quantities of timber are used underground, have complete installations of fire-service mains, generally $1\frac{1}{4}$ in., throughout the various workings, with tanks underground, placed, say, every 300 ft., to avoid excessive pressure on the mains, and kept full by means of ball float-valves.

The local experience with regard to underground fires has been that the only hope of subduing them is by the prompt application of a small quantity of water under pressure, and therefore the lighter the gear the more likely it is that the fire will be subdued before it reaches portions of the workings that might quickly become inaccessible through the burning out of the timbering. For this reason, $\frac{3}{4}$ -in. hose in 60-ft. lengths, placed in the levels every 200 ft., is used on the Proprietary mine.

With a view to preventing the spread of fires which may obtain too

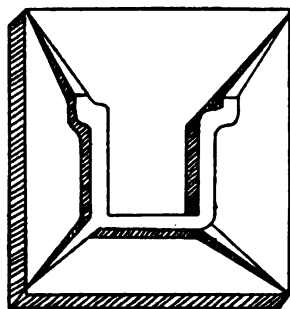


FIG. 15.—CASTING AT THE JUNCTION OF SET TIMBERS TO HOLD THE END OF A SPREADER.

strong a hold on square-set stopes to permit extinction without flooding the workings (which is sometimes impracticable), the Proprietary Company has made a practice of forming, at various portions of the workings, barriers to the progress of fires by having all timber connecting adjoining sets replaced with iron or steel spreaders, generally railway rails; and special plain castings (Fig. 15) are placed at the junction of the set timbers to hold the end of rail in position and prevent them entering the timber by increasing the area of base. If these barriers are constructed while ordinary stoping proceeds, very little added expense is incurred, as there is generally a stock of old rails available at low cost.

The various mines have provided themselves with different forms of smoke-jackets to enable the wearers to enter workings containing an irrespirable atmosphere, goggles being also provided to protect the eyes from smoke. Among those employed are the "Proto" and "Pneumatogen" types, using compressed oxygen, and the "Aerophor" using

liquid air; the last named has only recently been introduced at the North mine.

This apparatus is looked after by the fire station and ambulance staff, resident at each large mine, under the supervision of the underground manager, and in addition to the members of these staffs the shift bosses receive periodical practice with the breathing apparatus. Each of these mines possesses pulmotors, in the use of which the ambulance staff and others are trained, so that respiration can be most effectively restored in cases of "gassing," etc.

Underground Staffs

The problems to be faced in the working out of the large bodies of ore occurring in this district call for the application of scientific principles to a much greater extent than is the case with narrow lodes.

In the earlier days when the exploitation of the orebodies was not systematized to the extent that now obtains, creeps due to the collapse of stopes were frequent. On this account it has been the practice for a considerable number of years to put in charge of the underground work an officer who has received thorough scientific training as a mining engineer, combined with sufficient practical experience. In some cases this consisted of 18 months' manual work in various capacities underground, etc., in other cases, three or four times this length of service as underground surveyor. This officer usually has a similarly trained assistant, or an understudy is being trained while on the underground survey staff.

Under the underground manager and his assistant, there are day foremen, as many as necessary, according to the extent of the workings allotted to him. To these men fall the duties of letting all the ordinary ore-breaking and development-work contracts, besides looking after the general safety of the workings, arranging for the filling of stopes as required, and many other duties that it is not necessary to particularize.

Next, there are in the largest mines assistant foremen who are competent to take the place of the day foreman in case of his absence; and these men follow each shift, the one on the day shift generally having charge of the wages-timbermen who, with few exceptions, work on day shift only. Finally, there are the shift bosses who are picked men drawn from the ranks of the miners and who may ultimately become foremen. In larger mines there is a separate boss in charge of the truckers, both on ore and in filling stopes—the ordinary boss confining his attention to miners' work.

An important part of the duty of shift bosses is to see to the safety of the backs, etc., of the stopes. Familiarity with danger too often leads to contempt of it on the part of the miners, and consequently to accidents.

Timekeepers check the attendance of the workmen when entering and coming up the shaft, and make up distribution sheets showing the branch of work against which their earnings are to be charged, *i.e.*, stoping, development or preparatory work, repairs of different kinds, etc., as the case may be.

The contracts are let as a rule to parties of six men, *i.e.*, two per shift; but sometimes this number is doubled especially if separate chutes are not available. As a general rule, a small party works more amicably than a large one, and for this reason is preferable. The earnings are made up and distributed to the various individual members of the contracting party by the companies *pro rata*, according to the number of shifts worked, thus relieving the party of the necessity of accurately apportioning the proceeds of the contract.

Costs

Complete cost sheets are made up showing the cost per ton of the various items of expenditure, there being subdivisions into: (1) Labor; (2) Stores (including coal and water for power and all supplies); (3) Miscellaneous, and (4) Special expenses not directly chargeable to ore raising, but which must be included in the total cost.

At the Proprietary mine, complete costs are worked out and made available to the management two days after the close of the week; and, in addition to the complete weekly costs, daily cost sheets on somewhat less elaborate lines are also made up; most of the items in these daily costs are accurately stated, but in a few instances where it is difficult to obtain the daily quota, average figures are used.

The total cost of mining in the district varies considerably according to the nature of the ore and its mode of occurrence—whether in large regular masses or separate veins, or in pillars left between old stopes, or in isolated bodies. But, in general, it may be said that the costs range from 15 to 20 shillings per ton when, as during 1914, the earnings of miners on contract averaged about 17s. 6d. per 8-hr. shift.

In conclusion, I wish to acknowledge the assistance rendered by the underground manager of the Proprietary mine, and his assistants (Messrs. Slee and Coldham), and also the courtesy shown by the managers of the various other mines in facilitating the preparation of this paper.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Behavior of Stibnite in an Oxidizing Roast

BY H. O. HOFMAN,* BOSTON, MASS. AND JOHN BLATCHFORD,† OAK PARK, ILL.

(New York Meeting, February, 1916)

THE leading antimony mineral is stibnite. In smelting stibnite ore two processes are available, precipitation and roasting-reduction. The former is suited only for high-grade ores. As low-grade ores are more common than high-grade, roasting-reduction is of greater importance than precipitation. In the roasting process the aim may be to leave the oxidized antimony in the ore, or it may be to volatilize as much of the antimony as possible, collect the volatilized oxide as a rich intermediary product and smelt it for antimony, leaving the gangue poor enough to be considered a waste product.

Whichever way the roast is conducted certain difficulties inherent in stibnite are encountered. These are:

1. The low melting point of stibnite which, according to Pélabon,¹ is 550°C., according to Wagemann² 540°C., and according to Borgström³ 546°C.

2. The ignition temperature: According to Friedrich,⁴ stibnite, if heated in air, begins to oxidize at 290°C. if the size of a grain is 0.1 mm. in diameter; at 343°, if 0.1 to 0.2 mm.; and 430° if 0.2 mm.

3. The fusibility of a mixture of Sb_2S_3 and Sb_2O_3 which in the form of kermesite $(\text{Sb}_2\text{S}_3)_2 \cdot \text{Sb}_2\text{O}_3$ melts at 517°C.⁵

4. The volatility of Sb_2S_3 and Sb_2O_3 , for which no numerical data appear to exist, although practical experience has shown that they are volatile at low temperatures.

As regards the oxidation of metallic antimony, we have the experimental evidence of C. F. Plattner⁶ that when fused and brought to a red heat, it burns with a bluish-white flame to Sb_2O_3 which passes off as a

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¹ *Comptes rendus*, vol. cxxxvii, p. 920 (1903).

² *Metallurgie*, vol. ix, p. 518 (1912).

³ *Chemical Abstracts*, vol. ix, p. 2364 (1915).

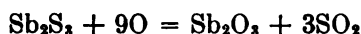
⁴ *Metallurgie*, vol. vi, p. 169 (1909).

⁵ Borgström. *Loc. cit.*

⁶ *Die Metallurgischen Röstprocesse*, Engelhardt, Freiberg, 1856, p. 162.

whitish fume, and that Sb_2O_3 can be further oxidized to Sb_2O_4 or Sb_2O_5 . If Sb_2O_4 is considered to be antimonious antimonate, the compound falls in line with metallic antimonates which are formed if a metallic antimonide or sulphantimonide such as pyragyrite, tetrahedrite, or jamesonite, is subjected to an oxidizing roast—or if Sb_2O_3 in contact with a finely divided metallic oxide is heated in a current of air.

The chemical change which takes place in roasting stibnite is usually expressed by the equation



but the Sb_2O_3 is further oxidized to Sb_2O_4 . In carrying on a roast, the temperature of the ore is held at first at about 350°C ., and the charge is rabbled more or less continuously to prevent or correct caking; later, when part of the Sb_2S_3 has been converted into Sb_2O_3 and Sb_2O_4 , the temperature may be raised with advantage. Roasting changes the black sulphide into a yellowish-white oxide.

The aim of the present investigation was to study the changes stibnite undergoes when roasted at different temperatures; to determine the composition of the roasted product; and to ascertain the losses which occur.

The material used was crude stibnite of unknown origin from the metallurgical collection of the Massachusetts Institute of Technology. It was a compact grayish crystalline mass showing fine prismatic needles. The fresh fracture appeared silver-white and had a metallic luster. The general appearance was not unlike that of graphite. After grinding, it had a jet-black color. A chemical analysis gave 71.66 per cent. antimony and 0.96 per cent. insoluble. Calculating the antimony as the sulphide gives 100.3 per cent. Sb_2S_3 ; the excess over 100 is probably due to the presence of a small amount of metallic antimony.

The roasting furnace used was of the electric resistance type. It consisted of a porcelain tube, 10 in. long and $2\frac{3}{4}$ in. in diameter, wound with "nichrome" wire; this was surrounded by mineral wool, and the whole inclosed in a pipe of galvanized iron, $7\frac{1}{2}$ in. in diameter. Temperatures were measured with a thermo-electric pyrometer.

The ore was ground to pass a 20-mesh sieve; a 5-gram sample was used, as this covered the bottom of the porcelain boat and filled it to about one-third of its depth.

During a roast the ore was frequently rabbled with a wire bent to the form of a hook. A piece of moistened litmus paper held above the surface of the ore served as a test for sulphur dioxide. Any odor, or change in the appearance of the ore was noted. The temperature of the furnace was raised gradually to the point at which it was desired to finish a roast, and held there until no more changes in appearance and weight of the charge occurred.

Seven experiments were carried out, covering a range of finishing temperatures from 337.5 to 545.4°C. Experiment No. 3, in Table I, shows the manner in which the records were taken.

TABLE I.—*Record of Roast No. 3*

Time	Galvano- meter, Deflection	Cold Junction, °C.	Tempera- ture, °C.	Current, Amperes	Remarks
11 : 15	0.0	22.5	22.5	3.0	Start.
11 : 30		22.5		4.0	
11 : 45		22.5		4.0	
12 : 00	7.3	22.5	133.0	5.0	
12 : 15	11.0	22.5	175.0	5.0	
12 : 30	12.5	22.1	192.0	6.0	Faint odor of SO ₂ . Litmus begins to turn pink very slowly.
12 : 45	15.0	22.1	220.0	8.0	
1 : 00	18.0	22.5	257.0	10.0	Litmus changes to pink slowly.
1 : 15	24.0	22.5	328.0	10.2	Litmus changes to pink quickly; odor of SO ₂ stronger.
1 : 30	23.2	22.5	315.0	9.0	Ore begins to look dull.
1 : 45	25.0	22.5	337.0	9.6	Ore is gray.
1 : 52	27.0	23.0	362.0	10.0	Very sharp odor of SO ₂ .
2 : 00	28.0	23.0	373.0	9.5	White ashes on rabbling, visible gas evolved.
2 : 15	27.5	23.0	368.0	9.6	Rabbling promotes formation of ashes.
2 : 30	28.0	23.5	374.0	9.6	
2 : 45	28.5	23.6	379.0	9.6	Odor of SO ₂ not so strong; white kernels and gray powder form.
3 : 00	28.0	23.6	374.0	9.5	Litmus changes to pink readily.
3 : 15	28.0	23.8	374.0	9.4	
3 : 30	27.5	24.1	369.0	9.5	
3 : 45	27.0	24.5	364.0	9.8	
4 : 00	27.5	24.8	369.0	10.0	Litmus changes to pink very slowly.
4 : 15	28.0	24.8	375.0	10.0	Litmus changes to pink very slowly.
4 : 30	28.1	25.0	376.0	9.0	No change in litmus.
4 : 45	27.5	25.0	370.0	9.5	No odor of SO ₂ . Stopped.

	Grams
Weight of raw ore.....	4.9346
Weight of residue.....	4.4354
Loss weight.....	0.4992
Average finishing temperature.....	372°C.
Color of ground residue.....	Gray

A summary of the observations made during the six roasts is given in Table II.

TABLE II.—*Summary of Observations in Roasting Experiments*

Temperature, °C.	Changes
196	Faint odor of SO_2 .
219	Litmus begins to turn pink.
281	Ore becomes dull; odor of SO_2 very distinct.
336	White ashes form on rabbling, and some Sb_2O_3 is evolved.
380	Lowest temperature at which odor of SO_2 disappears.
400	Ore becomes sticky if temperature is raised quickly.

The first observation made in each roast was the evolution of sulphur dioxide; this was followed by a change in the color of the litmus paper at a slightly higher temperature. At 280°C . the ore began to lose its luster and became dull. Then followed a strong odor of sulphur dioxide which directly preceded a sudden visible formation of Sb_2O_3 at 336°C . The oxide forms spontaneously at this temperature in the form of little yellow grains which become white upon cooling. As the temperature is raised above 336°C ., the grains seem to break up into a grayish powder. If the temperature is raised too quickly before all the sulphide has been changed into whitish grains of oxide, the charge becomes sticky; this is not the case if the temperature is raised gradually. In roasts Nos. 6 and 7 a small amount of Sb_2O_3 was deposited on the rim of the heating tube, but the temperature and time at which the deposit was formed were not noted. Tables III, IV, and V contain data on the roasting experiments and the results obtained.

TABLE III.—*Summary of Roasts*

Roast	Time, Hours	Raw Ore, Grams	Finishing Temperature, °C.	Loss of Weight, Grams	Loss of Weight, Per Cent.	Loss of Sb, Per Cent.
2	$6\frac{1}{4}$	4.9994	337.5	0.3520	7.05	None
3	$5\frac{1}{2}$	4.9346	372.0	0.4992	10.10	None
4	$3\frac{1}{2}$	5.0480	407.5	0.6267	12.40	2.21
5	$2\frac{3}{4}$	5.0382	454.5	0.7902*	15.70*	5.60*
6	$3\frac{1}{2}$	5.0028	493.5	0.5798	11.60	2.93
7	$4\frac{3}{4}$	5.0388	545.4	0.5963	11.80	4.21

* This excessive loss in weight is due to some experimental accident which cannot be definitely placed.

TABLE IV.—*Summary of Products*

Roast	Residue, Grams	Insoluble*		Sulphur, Per Cent.	Trivalent Antimony, Per Cent.	Total Antimony, Per Cent.	Sb ₂ S ₃		Sb ₂ O ₃		Sb ₂ O ₄	
		Per Cent.	Grams				Per Cent.	Grams	Per Cent.	Grams	Per Cent.	Grams
2	4.6474	0.96	0.0446	13.46	77.22	76.67	46.9	2.1760	52.7	2.424	None	None
3	4.4354	0.96	0.0426	2.86	80.76	80.33	10.0	0.4439	87.7	3.905	None	None
4	4.4213	0.96	0.0424	None	75.00	80.01	None	None	84.0	3.713	12.7	0.561
5	4.2480	0.96	0.0408	None	61.85	80.23	None	None	52.7	2.216	46.1	1.978
6	4.4230	0.96	0.0425	None	52.44	78.70	None	None	31.4	1.389	66.5	2.942
7	4.4425	0.96	0.0427	None	41.77	77.85	None	None	6.8	0.303	91.5	4.061

* Based on analysis of raw ore.

TABLE V.—*Percentage of Antimony Compounds in Roasted Products*

Run	Total		Based on weight of raw ore		
	Per Cent.	Grams	Sb ₂ S ₃ , Per Cent.	Sb ₂ O ₃ , Per Cent.	Sb ₂ O ₄ , Per Cent.
2	100.00	4.6476	43.5	48.5	None
3	99.04	4.2915	9.0	79.2	None
4	97.63	4.3164	None	73.6	11.1
5	99.86	4.2348	None	43.8	39.2
6	98.91	4.3735	None	27.8	58.8
7	99.40	4.4170	None	6.0	84.5

A graphical representation of the results, given in Fig. 1, shows clearly the changes which stibnite undergoes when it is roasted. No change takes place until the gradual rise in temperature has reached 200°C. when SO₂ begins to be given off. The quick fall of the Sb₂S₃ curve shows that beginning with 200°C. a rapid oxidation takes place which is terminated at 400°C. It is fair to assume that the formation of Sb₂O₃ commences at the temperature when SO₂ is first detected. The Sb₂O₃ curve is therefore started at about 200°C; it has a rapid rise which corresponds to the quick fall of the Sb₂S₃ curve. Its highest point coincides on the temperature scale with the point at which the oxidation of Sb₂S₃ is complete; the Sb₂O₄ curve starts at the same temperature. As the temperature rises, the Sb₂O₃ curve falls and is accompanied by a corresponding rise in the curve for Sb₂O₄. From this it is seen that in roasting Sb₂S₃ there is formed at first only Sb₂O₃, and that this is converted into Sb₂O₄ only when all the Sb₂S₃ has been oxidized. Thus the amount of Sb₂O₄ formed is dependent upon the temperature and the time of the roast. It is assumed that there is no loss of weight below the temperature at which SO₂ begins to be evolved, i.e., about 200°C. From this point on, the loss in weight increases with the temperature and the formation of Sb₂O₃; it reaches a maximum at about 450°C. then falls and afterward becomes

nearly constant. The loss in antimony appears to be a function of the temperature; the curve shows that it increases with the finishing temperature with the exception of the jog at 450°C. which, as stated with Table III, is an unexplained experimental accident.

The following conclusions may be drawn from the investigation:

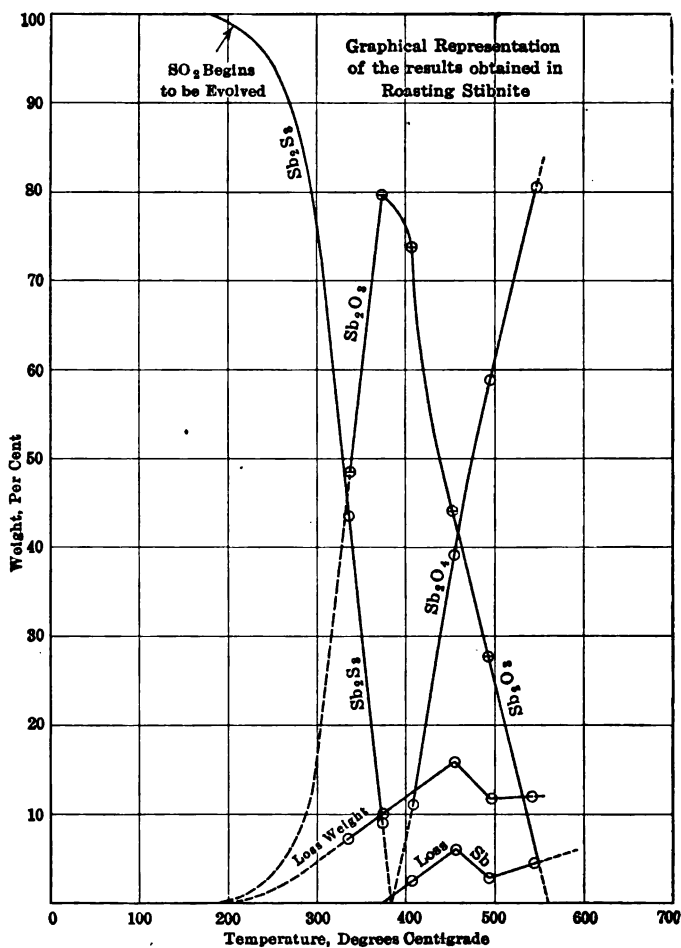
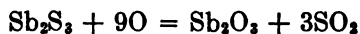


FIG. 1.—RESULTS OBTAINED IN ROASTING STIBNITE.

1. The ignition point of stibnite is approximately 200°C. as evidenced by the odor of SO_2 evolved and its action on litmus paper.

2. There is a sharp visible formation of Sb_2O_3 at 336°C. At this point white fumes are evolved.

3. The formation of Sb_2O_3 takes place slowly, probably according to the reaction:



4. It is possible to eliminate all the sulphur slightly below 400°C. without a large loss of antimony.

5. Stibnite is oxidized at first to antimony trioxide which begins to change to antimony tetroxide only when all the sulphide has been decomposed.

6. The amount of tetroxide increases as the trioxide decreases, the action being more rapid as the finishing temperature is raised.

7. The loss of antimony during the roast increases with the temperature.

A discussion of the chemical work is given in the paper by W. T. Hall and J. Blatchford, read at this meeting.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Determination of Antimony in the Products Obtained by Roasting Stibnite

WILLIAM T. HALL,* BOSTON, MASS. AND JOHN BLATCHFORD,† OAK PARK, ILL.

(New York Meeting, February, 1916)

THE product obtained by roasting stibnite is likely to contain some unoxidized antimony trisulphide and a mixture of antimony trioxide and antimony tetroxide. It was desired to determine, as accurately as possible, the condition of the antimony as well as the total quantity present. Attempts were made to separate the trisulphide and the two oxides by methods based upon their varying solubilities in different solvents, but no satisfactory results were obtained in this way. It was found, however, that by determining the total antimony content, the antimony present as trioxide, and the antimony present as trisulphide, a good idea of the chemical composition of the roasted product could be obtained.

The total antimony was determined by dissolving the sample in concentrated hydrochloric acid, reducing the antimony entirely to the trivalent condition by means of hydriodic acid and eventually titrating the antimony back to the pentavalent condition by means of iodine in the presence of sodium bicarbonate.¹ The details of the procedure are as follows:

Weigh out 0.25 gram of the roasted product into a trapped flask (Fig. 1), add 4 grams of powdered tartaric acid, 2 grams of potassium iodide and 40 c.c. of concentrated hydrochloric acid. Boil the solution gently for 5 min. and then cool to room temperature by shaking the flask while holding it under running water. Then, very carefully discharge the iodine color by the cautious addition of 0.05-normal sodium thiosulphate solution, adding a little starch toward the last. Nearly neutralize the acid with ammonia solution, but leave the solution distinctly acid. Pour the slightly acid solution into 200 c.c. of water con-

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† Analytical Chemist and Metallurgist.

¹ F. A. Gooch and H. W. Gruener: *American Journal of Science*, Series 3, vol. xlii, p. 213 (1891).

taining an excess of sodium bicarbonate and titrate to a permanent blue color with standard 0.1-normal iodine solution.

The determination of the unchanged antimony sulphide was accomplished by a method corresponding to that used in the determination of sulphur in steel.² The sample was dissolved in concentrated hydrochloric acid, the escaping hydrogen sulphide absorbed in an ammoniacal cadmium chloride solution and the sulphur in the precipitated cadmium sulphide determined iodometrically.

Inasmuch as all the antimony present in the roasted stibnite was soluble in concentrated hydrochloric acid, the solution remaining in the evolution flask after the determination of the sulphur could be used for the determination of the trivalent antimony. The details of the procedure are as follows:

To the solution remaining in the evolution flask after the determination of the sulphur, add 4 grams of powdered tartaric acid and carefully dilute with 100 c.c. of water. Carefully add 6-normal ammonia

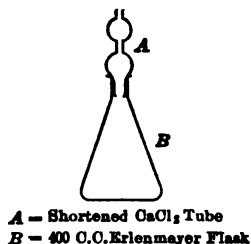


FIG. 1.—APPARATUS USED IN DETERMINATION OF ANTIMONY IN ROASTED STIBNITE.

solution until the solution is only slightly acid and then pour the solution into a large beaker containing 5 grams of sodium bicarbonate in 200 c.c. of water. Titrate with 0.1-normal iodine solution as in the determination of the total antimony.

In computing the results, it is necessary to remember that antimony tetroxide may be regarded as antimonious antimonate:



Since antimony pentoxide changes to the tetroxide when heated above 300° , it is fair to assume that each atom of pentavalent antimony found in the analysis corresponds to one molecule of antimony tetroxide. Then from the total quantity of trivalent antimony found, deductions must be made for the amount of tetroxide and for the trisulphide as found by the sulphur determination; the balance is the trivalent antimony corresponding to the quantity of antimony trioxide present.

Comparatively little information concerning the chemical properties

² A. A. Blair: *The Chemical Analysis of Iron*, 7th Edition, p. 60. H. Kinder: *Stahl und Eisen*, vol. xxviii, p. 249 (1908). Massenez: *Ibid*, vol. xxxii, p. 2089 (1912)

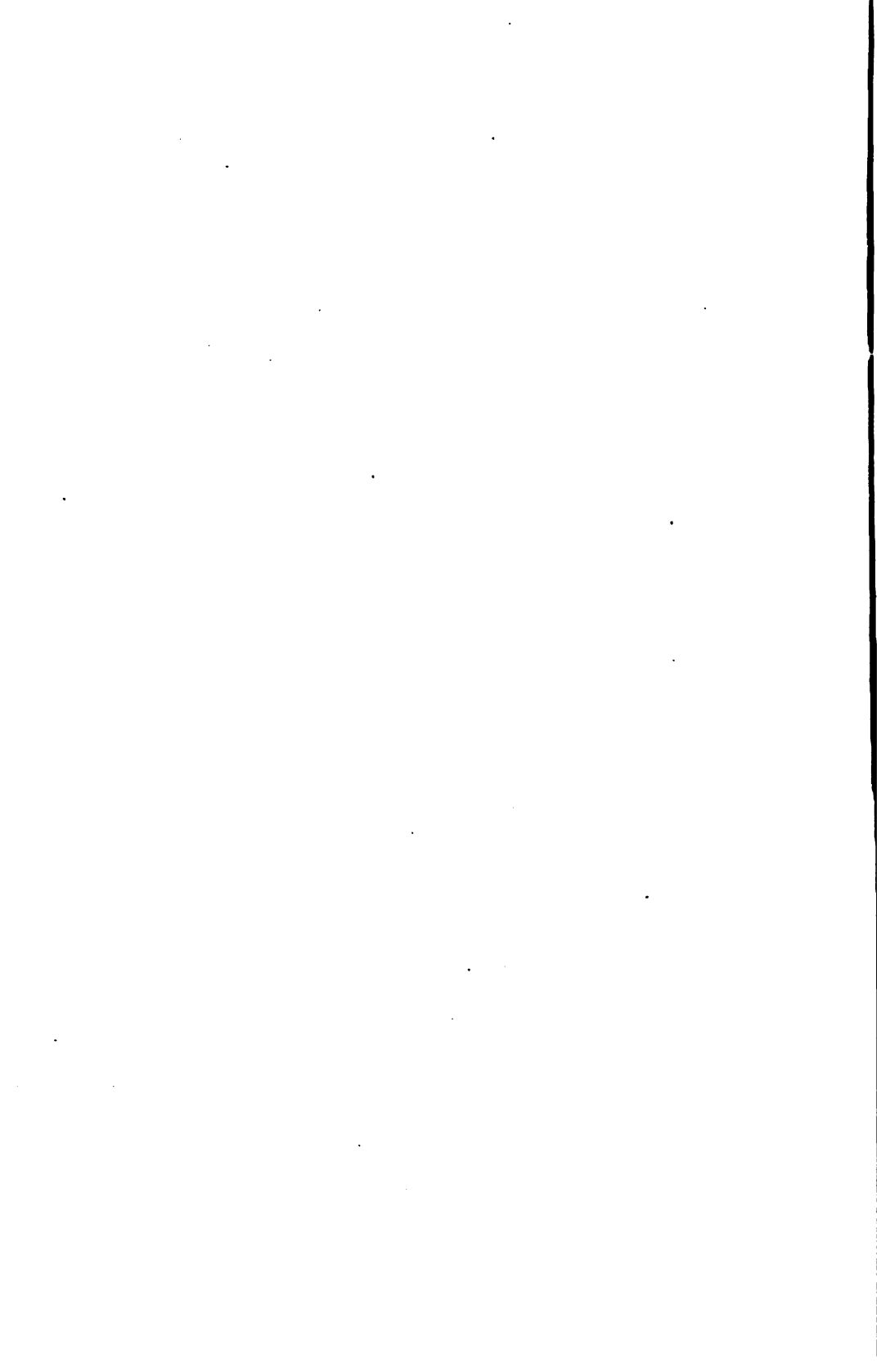
of antimony tetroxide is to be found in the larger treatises on inorganic chemistry. From the literature, it was expected that considerable difficulty would be encountered in dissolving antimony tetroxide in hydrochloric acid and it was surprising, therefore, to find no particular trouble from this source. Again, it was to be expected that in dissolving the roasted product, when some unchanged sulphide was present, some reduction of the pentavalent antimony and corresponding oxidation of the hydrogen sulphide to free sulphur would occur.

The accompanying table gives a summary of the results obtained in the analysis of the roasted stibnite.

Analyses of Roasted Stibnite

	Sulphur, Per Cent.	Trivalent Antimony, Per Cent.	Total Antimony, Per Cent.
Run II.....		77.50*	76.60
	13.47	77.47*	76.74
	13.32	76.68	
Average.....	13.40	77.22	76.67
Run III.....		80.79	81.25
	2.803	80.50	79.41
	2.918	80.98	
Average.....	2.860	80.76	80.33
Run IV.....	None	75.04	80.04
		74.96	79.97
Average.....		75.00	80.01
Run V.....	None	62.01	80.41
		61.69	79.54
			80.04
Average.....		61.85	80.23
Run VI.....	None	52.46	78.65
		52.41	78.74
Average.....		52.44	78.70
Run VII.....	None	41.75	77.96
		41.79	77.73
Average.....		41.77	77.85

* The values obtained when sulphide was present were a little high for trivalent antimony and a little low for sulphur.



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**A Development of Practical Substitutes for Platinum and its Alloys,
with Special Reference to Alloys of Tungsten and Molybdenum***

BY FRANK ALFRED FAHRENWALD, CLEVELAND, OHIO

(New York Meeting, February, 1916)

I. INTRODUCTORY

METALLURGICAL research has discovered many an alloy possessing properties not combined in any single metal, and progress still consists chiefly in the investigation and utilization of alloys. In the case of iron, the demands of automobiles, high-speed machines and high-duty engines have led to the production of special iron alloys which will meet any reasonable specifications in that field. In like manner, the bronzes, brasses and other alloys of copper have been brought to remarkable perfection, and for nearly every industrial purpose some alloy has been found more suitable than the pure metal.

Less complete success has attended the attempt to find substitutes for gold, platinum, and the other precious metals. Indeed, it is not likely that a material can be produced which will possess all the properties of any one of them; yet it is reasonable to hope that, for any given use, the properties required may be found in some less expensive material. Thus, in incandescent electric lamps, the wire passing through the thick glass neck of the bulb was, until recently, almost universally made of platinum, for the single reason that no other known material, suitable as a conductor, had the same coefficient of expansion as glass. But a comparatively recent investigation of the iron-nickel series¹ has shown that alloys of those metals may be produced, the coefficient of expansion of which can be accurately controlled between that of iron or nickel and zero. Thus, in that particular industry, a substitute for platinum has been found.

According to statistical reports, not only considerable quantities of gold and iridium, but also more than one-third of the annual supply of platinum, are used (and, in the nature of the case, irrevocably lost) by dentists. Platinum is thus employed in several forms. As a thin foil, it serves various purposes for which its high melting point, pliability, chemical resistance, and other properties—including the ease with which

* Condensed from a dissertation submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy in the University of Michigan.

¹ C. E. Guillaume: Non-expansive Alloys, *The Metallographist*, vol. vi, p. 162, 1903. Grenet and Charpy: Dilation of Steels at High Temperatures, vol. vi, p. 238, 1903.

it may be soldered—are invaluable. But it is most extensively used in the alloy with iridium, which is more resistant chemically than pure platinum, solders as readily, and possesses, besides, the quality of stiffness, and is not seriously softened by annealing at ordinary soldering temperatures. These advantages dictate its use, in spite of its high cost. The discovery of a substitute in dentistry for platinum and platinum-iridium, especially if it possessed useful properties which they lack, would be eagerly welcomed, and would find wide application in other arts also. The solution of this problem has been undertaken by the Research Foundation of the National Dental Association, for which the work described in this paper was done by the writer.†

The substitute desired must satisfy the following conditions:

1. Its melting point must be high, at least well above $1,200^{\circ}\text{C}$.
2. It must not be affected by those chemical compounds formed in its application, nor should it oxidize at a soldering temperature.
3. It must possess sufficient strength to resist stresses tending to change its form while in place, and at the same time be sufficiently pliable to be worked to the desired shape.
4. Its coefficient of expansion must be low, in order that desired dimensions may be easily produced in the finished product. (This factor is important, since the range through which this material is manipulated is often more than $1,000^{\circ}\text{C}$.)
5. It should unite readily with gold, silver and similar metals, and their solders.
6. Its cost of production should be low, as compared with that of platinum.

After the entire list of metals had been considered with respect to these conditions, it was evident that any search for the desired material must be among the alloys, since experience has shown that the physical properties of a metal may be radically changed by the addition of varying amounts of another element, or of several elements, as in the case of steels, brasses and bronzes.

Considerations based upon the periodic law of atomic weights, and the table constructed in accordance therewith by Mendeléeff and Lothar Meyer, led to the conclusion that the field for profitable research was narrowed to the elements chrome, manganese, iron, cobalt, nickel, copper, silver, palladium, gold, molybdenum and tungsten, and their alloys (ruthenium, iridium and osmium, likewise theoretically indicated, being ignored because of their costliness and scarcity).²

† This work was done in the Department of Mining and Metallurgy, Case School of Applied Science, and the Department of Chemical Engineering, University of Michigan.

² [NOTE.—The original thesis contains an interesting discussion of the theoretical considerations above mentioned. It assumes the probability that the physical properties (such as the melting points) as well as the chemical (such as base-forming or acid-forming) properties of the elements bear certain relations to their arrangement according to the periodic law. This discussion has been omitted here to save space.]

When two or more metals are brought together in the liquid state, they mix exactly like two ordinary liquids. When the temperature is lowered the solidified mass may contain any one of the four following constituents: Pure components; solid solutions; compounds; and eutectics—or some combination of these.

A comparison of the properties of different alloys containing these constituents has shown that they impart their characteristic properties to the alloy of which they form a part; in fact, the relation between the constitution of an alloy and its mechanical properties is so clearly defined that the possibilities of industrial application may be predicted for a given alloy, if its constituents are definitely known. Conversely, if a certain application is desired, as in the problem under consideration, a definite limit may be placed upon the number and amount of constituents permissible.

Fortunately, the number of constituents is limited to four, as given above. Pure metals impart their own characteristics; solid solutions are, in general, the ductile constituents (if formed of ductile metals or of a preponderance of one ductile metal); compounds and (usually) eutectics are hard and brittle, while the latter, even when present in very small amounts, tend to solidify between the grains of the alloy, thus destroying its ductility.

The problem was to determine what combination, if any, of the elements named, in whatever form, would meet the assumed set of specifications.

Accordingly, each of the 11 elements was combined, in varying proportions, with each of the other 10, giving 55 binary series to be considered with regard to their suitability as practical substitutes for platinum and platinum-iridium alloys.

II. WORK ON ALLOYS MADE BY FUSION

Behavior under the hammer, or when drawn through draw plates, and under the influence of acids and alkalis, was sufficient to indicate whether any particular specimen should be discarded at once or made the subject of further investigation.

It was not considered necessary to measure the various melting temperatures encountered, as only a complete fusion, free from all contaminants, was desired. The purity of all components was the highest obtainable, while the purity of the resulting alloys was checked by microscopical and, if necessary, by chemical analysis.

1. *Apparatus*

The Gran-Annular electric furnace³ was used in this set of experiments, and served as no other type of furnace would have done under

³ Zay Jeffries: Notes on the Gran-Annular Electric Furnace, *Metallurgical and Chemical Engineering*, vol. xii, p. 154, 1914; also, C. H. Fulton: *Trans.*, vol. xlv, p. 769 (1912).

conditions involving constant use, quick heating and cooling, sensitive control, and low upkeep cost. This is of the granular-carbon resistance type, in which temperatures are limited only by the melting point of the



FIG. 1.—THE GRAN-ANNULAR ELECTRIC FURNACE.

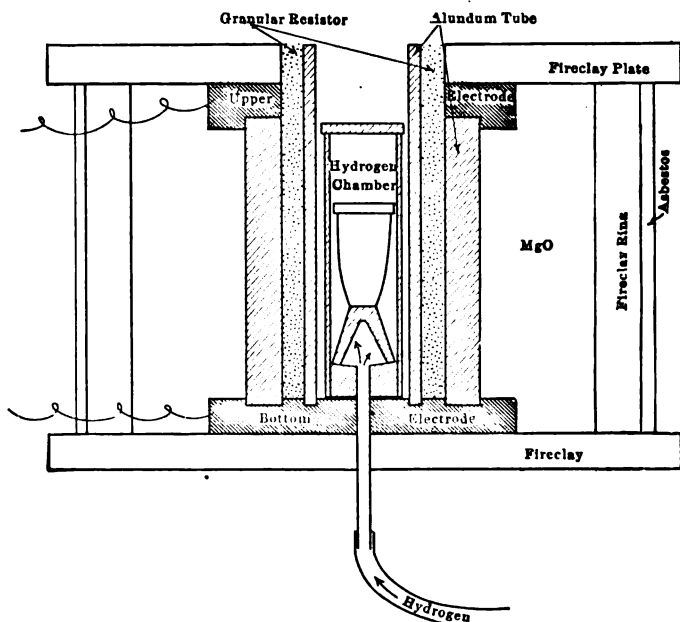


FIG. 2.—VERTICAL SECTION OF FIG. 1.

tubes used. With tubes of alundum, it can be safely run up to 1,800°C. This furnace is shown in Figs. 1 and 2, the latter a vertical section, showing also the means adopted for introducing any desired atmosphere into the crucible chamber. This latter device consists of a cylindrical alun-

dum crucible, fitted with a Marquard tube which passes through the bottom of the furnace and thence to the generator, provided with close-fitting cover, so that a gas may be maintained within it, and surrounding the crucible containing the melt, at a pressure sufficient to cause an outward flow through all the pores of the chamber walls, thus effectually preventing contamination by the atmosphere of the outer furnace chamber.

When this furnace is run above $1,200^{\circ}\text{C.}$, the atmosphere within the heating chamber is of the following analysis: O, 0.20 per cent.; N, 68.90; CO_2 , 0.70; CO, 30.20 per cent.

TABLE I.—*Materials, Etc., Employed in Experiments*

Metal	Crucible Material	Protective Cover	Inert Atmosphere
Chromium	Carbonaceous material not permissible.		H_2 or N_2 .
Molybdenum . . .	Magnesia may be used.		
Tungsten	Alundum questionable.		
Manganese	Magnesia, or a lining of magnesia. Alundum questionable. Silica or carbon must not be present.	BaCl_2 .	H_2 or N_2 .
Iron	Porcelain, magnesia, graphite with magnesia lining, alundum. Carbon or silica should not be in contact with fusion.	BaCl_2 . Gypsum.	H_2 . N_2 forms nitride. CO or CO_2 will carbonize.
Cobalt	Same as for iron, except that porcelain is strongly attacked by oxide formed.	Same as for iron.	H_2 or N_2 .
Copper	Charcoal, graphite, porcelain, fire-clay, magnesia, silica, alundum.	KCl plus NaCl, borax, K ₂ CO ₃ , charcoal.	H_2 , N_2 , CO. CO_2 , illuminating gas.
Silver	Nickel, iron, fire-clay, graphite, porcelain, alundum.	KCl plus NaCl, borax, KCu, charcoal.	Same as for copper.
Gold	Practically any non-metallic crucible.	None necessary.	None necessary.
Palladium	Magnesia, alundum. Carbonaceous material should not be present.	Glass.	H_2 forms alloy with palladium, although at lower temperature.

The tests for the general properties of the alloys, such as malleability, ductility, corrodibility, etc., are so simple as to require ordinary laboratory appliances only.

Table I gives a list of crucible materials, protective atmospheres, and covers, which have been found satisfactory in fusing these metals.

2. Results

Some of the 55 binary series, as, for instance, those high in iron, manganese, or chromium, indicated at once their inability to meet specifications, while others, as those of nickel-palladium, nickel-tungsten, etc.,

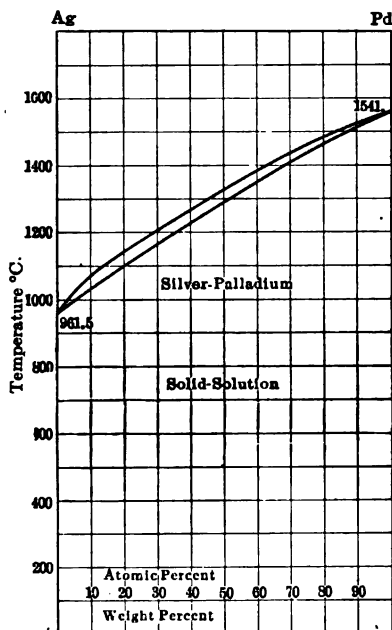


FIG. 3.—EQUILIBRIUM DIAGRAM OF THE SILVER-PALLADIUM SERIES.

required careful tests, and many degrees of concentration, to determine their comparative properties.

The investigation of these series (with the exception of those high in tungsten or molybdenum, which are not adaptable to fusion methods) is represented by nearly a thousand melts, which are unequally distributed among the different series, the number of fusions necessary in each series depending upon how nearly it approached the standard specifications.

The numerous negative results are not reported here, because they would greatly increase the volume, without correspondingly adding to the value, of this paper. It is sufficient to say that of these 55 series, only those of palladium with gold and silver have proved practically valuable.

The equilibrium diagram, Fig. 3, of the silver-palladium series⁴ shows a complete series of solid solutions. Alloys of these, in any proportions, have been found to be very soft, chemically inert, and if chosen of a composition to give a sufficiently high melting point, meet the necessary requirements of a foil to replace platinum, in the field of dentistry. The cost of this material will range from about 1 to 50 per cent. of that of platinum, depending upon the melting point desired.

The palladium-gold alloys (Fig. 4), however, are superior to those of silver-palladium, as gold is superior to silver. As in the palladium-silver series, the range between liquid and solid is very narrow, resulting in little

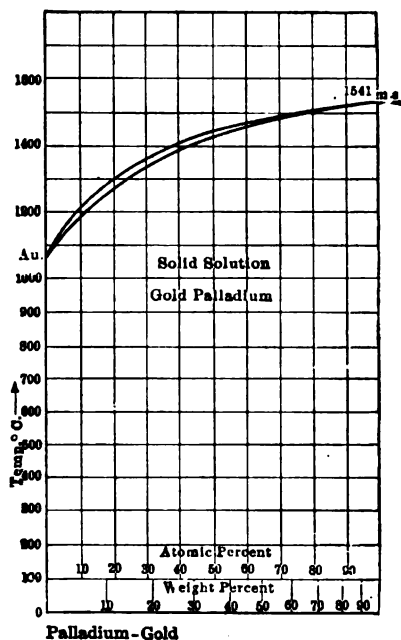


FIG. 4.—EQUILIBRIUM DIAGRAM OF THE GOLD-PALLADIUM SERIES.

tendency toward segregation. Fig. 5 shows curves having melting points as ordinates and cost per cubic centimeter, of silver-palladium and gold-palladium alloys, as abscissæ.

Metals are usually price-listed by weight; consequently an erroneous idea is generally prevalent as to the actual relative cost of different available materials, when applied to a given operation. A certain vessel, for instance, will require a definite volume of material, irrespective of its weight, which depends upon its density. Suppose, for example, that the volume of this vessel be 1 c.c. To fill this with platinum will cost, at

⁴Reur: Ueber die Legierungen des Palladiums mit Silber, *Zeitschrift für Anorganische Chemie*, vol. li, p. 315 (1906).

present quotations, about \$30.50; with gold, \$12.50; and with palladium, about \$22. This is not the popular conception of the relative cost of these elements which, when compared by weight, is gold, \$20.67; platinum, \$48; and palladium, about \$60 per ounce.

In connection with this series is a phenomenon for which no explanation is evident; 1 per cent. of palladium darkens gold perceptibly; at from 2 to 3 per cent., the color is a dull light bronze; while at about 10 per cent., no trace of a yellow color can be detected.

In the case of the silver-palladium alloys, a yellow tinge is perceptible

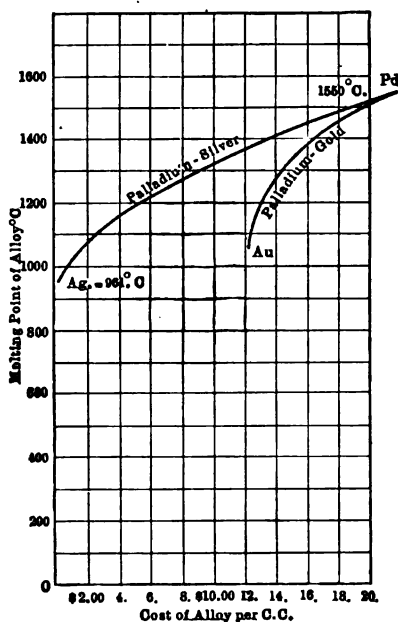


FIG. 5.—CURVES SHOWING RELATIVE COST OF PALLADIUM-SILVER AND PALLADIUM-GOLD ALLOYS HAVING EQUAL MELTING POINTS.

throughout the series, which phenomenon is equally hard to explain, considering the pure whiteness of both components.

Although platinum, in its softer forms, as in foil, may be replaced by the above-described palladium alloys, investigation to this point had produced nothing in the nature of a hard, strong, non-oxidizable, non-corrodible material which would serve as a substitute for the platinum-iridium alloys.

III. TUNGSTEN AND MOLYBDENUM

The investigation thus far had considered, both singly and in binary combination, the entire list of available metals with the exception of tungsten and molybdenum, the treatment of which, by ordinary fusion methods, was not feasible.

It may be said in advance that any treatment described in this chapter as applying to tungsten will apply likewise to molybdenum. Also, an outline of the properties of tungsten will include those of molybdenum, which will not be given in detail. Molybdenum is not so hard, not so stiff, and not so chemically resistant as tungsten; but its properties parallel those of tungsten so closely as to permit the omission of their detailed description.

1. *Properties of Tungsten and Molybdenum*

Tungsten is popularly best known in its application in the manufacture of incandescent lamps, and more technically, in the form of its alloys with iron and other metals forming special steels. Its alloys with cobalt, chromium, and similar metals, forming alloys of the "Stellite" type,⁵ have also found application.

The remarkable properties of the pure, metallic, ductile tungsten are, however, continually enlarging its field of application. This material is practically insoluble⁶ in any of the common acids; its melting point is higher than that of any other metal, its tensile strength exceeds that of steel; it is para-magnetic; it can be drawn to smaller sizes than any other metal, and its specific gravity is 70 per cent. greater than that of lead.

Wrought tungsten has been substituted with success for platinum and platinum-iridium, as contact points in spark coils, voltage regulators, telegraph relays, and for similar purposes. By reason of its greater hardness, higher heat conductivity, and lower vapor pressure, it gives much longer service than platinum. In the automobile industry, for instance, platinum has been in many cases entirely replaced for this purpose by tungsten.

Tungsten gauze is used successfully for filtering acid liquors and where fumes are encountered,⁷ while acid-proof dishes and tubes are also made of tungsten.⁸

Its tensile strength varies from 300,000 to 650,000 lb. per square inch; its thermal conductivity is more than twice that of platinum; its hardness varies from 4.5 to 8.0 (Mohr's scale), and its thermal coefficient of expansion is only 4.3 by 10^{-6} . That for molybdenum is yet

⁵ Described by C. L. Sargent: *Journal of the American Chemical Society*, vol. xxii, p. 783 (1900); and by E. Haynes: *Iron Trade Review*, vol. li, p. 927 (1912).

⁶ W. E. Ruder: Solubility of Wrought Tungsten and Molybdenum, *Journal of the American Chemical Society*, vol. xxxiv, p. 387 (1912).

⁷ C. Ehrenfeld: A Study of the Chemical Behavior of Tungsten and Molybdenum and Their Oxides, *Journal of the American Chemical Society*, vol. xvii, p. 381 (1895).

⁸ C. G. Fink: *Transactions of American Electrochemical Society*, vol. xvii, p. 229 (1910); and the *Proceedings of Eighth International Congress of Applied Chemistry*, vol. xxvi, p. 503 (1912). Also, I. Langmuir: *Transactions of American Electrochemical Society*, vol. xx, p. 237 (1911); and W. D. Coolidge: *Transactions of the American Institute of Electrical Engineers*, vol. xxix, Part ii, p. 961 (1910).

lower, 3.6 by 10^{-6} , while for platinum this value is 8 by 10^{-6} . For many other metals this ranges above 13 to 14 by 10^{-6} .

In short, ductile tungsten meets all but two of the preliminary set of requirements. The exceptions are, that it oxidizes easily at a red heat, and that it does not solder with gold and its alloys, except under strongly reducing conditions. Moreover, it was found that the larger-sized wires of this material were quite brittle, and even those sizes suitable for the purpose specified contained treacherous spots.

But the field of possibilities had, at this stage, narrowed itself to a consideration of this metal, with molybdenum as a second choice. It was therefore necessary to find: First, some means of preventing its oxidation; second, some means of increasing its affinity for gold, silver and their alloys; and third, some method of producing it in more reliable form.

2. Coating with Precious Metals

In earlier work on the gold and palladium series, it was discovered in attempting to alloy tungsten with these metals that while the larger pieces of tungsten were not appreciably dissolved, they were nevertheless removed from the molten bath with a beautiful impervious coating. As either of these furnished the surface qualities which tungsten lacked, the two remaining conditions seemed to be fulfilled. And so they were; but after this bath the tungsten wires were as brittle as glass.

TABLE II.—*Experiments in Coating Tungsten with Gold, Silver, and Palladium, under Varying Conditions*

Composition			Time Sec- onds	Temperature of Bath							
Per Cent.				1,100°C.		1,350°C.		1,500°C.		1,600°C.	
Ag	Au	Pd		Effect on Tungsten	Coat- ing	Effect on Tungsten	Coat- ing	Effect on Tungsten	Coat- ing	Effect on Tungsten	Coat- ing
100			10	None...	Very poor.	None...	Very poor.	Very brittle.	Poor.	Very brittle.	Poor.
100			30	None...	Very poor.	Quite brittle.	Very poor.	Very brittle.	Poor.	Very brittle.	Poor.
	100		10	None...	Good.	None...	Good.	Slightly brittle.	Good.	Very brittle.	Good.
	100		30	None...	Good.	None...	Good.	Quite brittle	Good.	Very brittle.	Good.
		100	10							Very brittle.	Good.
		25	10			Quite brittle.	Good.	Very brittle.	Good.	Very brittle.	Good.
		50	10					Very brittle.	Good.	Very brittle.	Good.
	50	50	10					Very brittle.	Good.	Very brittle.	Good.
	75	25	10					Very brittle.	Good.	Very brittle.	Good.
	90	10	10			None...	Good.	Very brittle.	Good.	Very brittle.	Good.
	95	5	10			None...	Good.	Very brittle.	Good.	Very brittle.	

These tests were made on drawn wires of 30-mil size, which without the coating, and heated in an atmosphere of hydrogen, crystallised and became brittle at 1,700°C. in from 10 to 20 min.

The problem thus arose of applying this coating, of such material, and under such conditions, that the other valuable properties of the tungsten would not be destroyed during the process. The crystallization of the tungsten was obviously due to one of three, or a combination of three causes; improper coating temperature, improper composition of the coating material, or improper time element.

In order to determine whether brittleness was due to the metal of the bath, or to the factor of time or temperature, tests were made, using varying composition and temperature of bath, and allowing the material to remain in the bath for varying periods.

Typical results, taken from the notes on these experiments, are given in Table II.

As shown in Table II, pure gold gives the best results. Both silver and palladium seem to accelerate crystallization, for some reason not

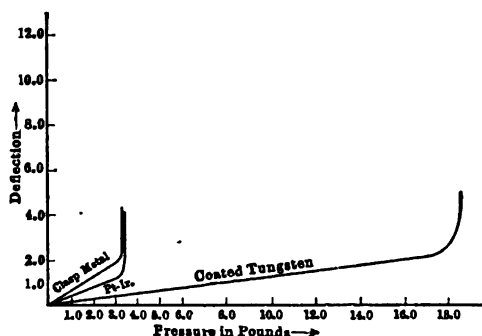


FIG. 6.—DIAGRAM SHOWING COMPARATIVE STIFFNESS OF COATED TUNGSTEN, "CLASP METAL," AND PLATINUM-IRIDIUM ALLOY CONTAINING 30 PER CENT. IRIIDIUM.

apparent at the present time. Silver has practically no affinity for tungsten, under ordinary conditions; gold will take up small amounts with difficulty; while palladium will readily dissolve certain amounts of it.

Pure gold, alone, forms a beautiful adherent coating which would serve in ordinary cases; but by adding small amounts of palladium to the coating bath, its melting point is raised, and, at the same time, a better bond between the tungsten and this protective layer is obtained.

Most of this coating was performed in an open atmosphere. The tungsten was first dipped into a bath of fused sodium carbonate, which operation caused a cleansing and protective shell of this material to cling to the specimen when it was withdrawn. This flux was automatically removed when the tungsten was dropped into the metal bath.

Properties of the Coated Material.—The material produced in this manner presents the surface properties of pure gold, or its alloys, together with the original internal properties of pure tungsten, a combination

which cannot be approached by any other metal or alloy at present available. With few exceptions, the alloys of platinum and iridium are inferior to this coated tungsten.

This is because of the great strength and stiffness of this new material, and due to the fact that it can be soldered, or otherwise manipulated, at high temperatures, without softening or losing this stiffness and elasticity to the slightest degree.

A comparison of the stiffness of coated tungsten with that of similar test pieces, one of which is an alloy containing 70 per cent. of Pt, and 30 per cent. of Ir, and the other the so-called "clasp-metal," an alloy of gold, copper and platinum, is shown in Fig. 6. In making these experiments the test wires were supported upon two knife edges 0.5 in. apart, weight being applied upon a third, midway between these two.

The chief remaining objection to this material is its frequent brittleness and unreliability in the larger-sized (above 40 to 50 mils) wires. The quality of this material, as supplied, however, is constantly improving. A discussion of the probable causes of this brittleness, and a suggested method for improvement are given below.

IV. ALLOYS OF TUNGSTEN AND MOLYBDENUM

1. *The Production of Ductile Tungsten and Molybdenum.*

The extreme brittleness of tungsten and molybdenum, when produced from the molten state, seems also to be an unavoidable characteristic of their high percentage alloys, when produced in a similar manner. However, in view of the remarkable success which has been attained in the production of these metals in the ductile form, by using methods not involving a preliminary molten condition, and because of the remarkable properties of these metals when so produced, it seemed advisable to determine whether the few undesirable properties could not be properly modified by introducing varying small amounts of other elements, using methods similar to those employed in the production of pure metal.

It was thought that perhaps the addition of small amounts of molybdenum would decrease the extreme brittleness of tungsten, and perhaps result in a material more pliable and ductile in the larger masses, or that the presence of small amounts of the more noble metals would lessen its tendency to oxidize; perhaps even prevent oxidization below relatively high temperatures, and at the same time add such surface qualities that it could be readily brazed or soldered with gold and other precious metals, under ordinary atmospheric conditions.

A description of the manufacture of ductile tungsten, as taken from the literature and patent specifications, gives the operation essentially as follows: The pure tungstic oxide, in certain cases containing a small percentage of ThO_2 (the effect of which will be discussed later), is reduced

in an atmosphere of hydrogen. This reduced powder is compressed into briquets, about 0.5 by 0.5 by 15 cm. in size, which are first sintered at about 1,300°C. and then heated electrically to a temperature near the melting point of the tungsten; after which, by successive swagings at temperatures above a red heat, the material is compacted and welded to a solid metallic mass, and when reduced to about 30 to 40 mils in diameter, it is drawn through diamond or ruby dies, first hot, and finally cold. All operations involving temperatures above a very dull red are performed in an atmosphere of hydrogen, or nitrogen.

No reference has been found describing any sort of metallographic control in connection with these various operations, and so far as available information indicates, the above method for the production of ductile tungsten and molybdenum has been developed and employed without assistance from this branch of science. This will, no doubt, account for the difficulty which has apparently been encountered in adapting it to the production of the material of larger sizes in ductile form, and in eliminating the last traces of brittleness in the drawn wires.

No manufacturer of ductile tungsten has been able, as yet, to supply our laboratory with specimens of ductile high-percentage alloys of tungsten. Directly because of this fact, and with confidence in the final success of a method involving proper metallographic control during the various stages of manufacture, experiments were begun to determine the conditions under which such alloys might be produced.

2. The Internal Structure of Metals⁹

Pure metals, in the solid state, are aggregates of crystals. These do not as a rule possess any regular geometrical shape, but the individual crystalline grains forming this aggregate possess the essential character

⁹ Sir J. Alfred Ewing: The Inner Structure of Simple Metals, *Journal of the Institute of Metals*, viii, p. 4 (1912).

Ewing and Rosenhain: The Crystalline Structure of Metals, *Philosophical Transactions*, vol. xciii, p. 353 (1900); vol. ci, p. 279 (1901).

J. E. Stead: The Crystalline Structure of Iron and Steel, *Journal of the Iron and Steel Institute*, vol. i, p. 145 (1898).

See also the following for relevant matter:

R. Hooke: *Micrographic*, London, 1665.

Reaumur: *L'Art der convertu le fer forge en acier*, Paris, 1722.

Sorby: *British Association Report*, 1864.

Osmond and Werth: Des Propriétés de l'Acier, *Annales des Mines*, Series 8, vol. viii, p. 6 (1885).

M. F. Osmond: Sur la Cristallographie du fer, *Annales des Mines*, Series 9, vol. xvii, p. 110 (1900).

James Thomson: *Philosophical Trans.*, Glasgow (1882).

Behrens: *Das Mikroskopische Gefüge der Metalle und Legierungen*.

Lehmann: *Molekularphysik*, vol. i, p. 279.

of all crystals, in having a regular arrangement, or orientation, of matter within their boundaries.

Fig. 7 is a photograph of the cast surface of pure gold, which has been slowly and carefully cooled, showing how the interior orientation of molecules has influenced the configuration of the surface of each grain. This mass of metal is seen to be made up of two or three crystal groups.

In Fig. 8, the surface is seen to be composed of a larger number of separate grains, irregular both in size and shape. This is a photograph of a more quickly cooled specimen. In this the grains are distinguished not only by these irregular boundaries but by a difference in texture between one grain and another; some are bright, some are dark, while others range



FIG. 7.—CAST SURFACE OF PURE GOLD SLOWLY COOLED. $\times 6$ DIAMETERS.

intermediately. If the source of light used in making this micrograph be moved, the brightness of the individual grains will change, as shown in Fig. 9; those which appeared bright or dark under the first condition of lighting will assume a different shade when illuminated from a different angle. This change is due to the fact that the surface exposes a multitude of little facets, or planes, all facing one way in any one grain, but of a different inclination in the different grains. The brightness of each grain depends upon the relative angles of these reflector systems.

Surface conditions indicate that the entire volume of any one grain consists of an assemblage of structural units, which may be likened to the bricks of a wall, all parallel in any one grain but facing differently in the different grains. This has been further verified by a study of cross-sec-

tions of metals, and by observation of the fact that certain crystals have been found to act as a three-dimension diffraction grating toward x -rays.

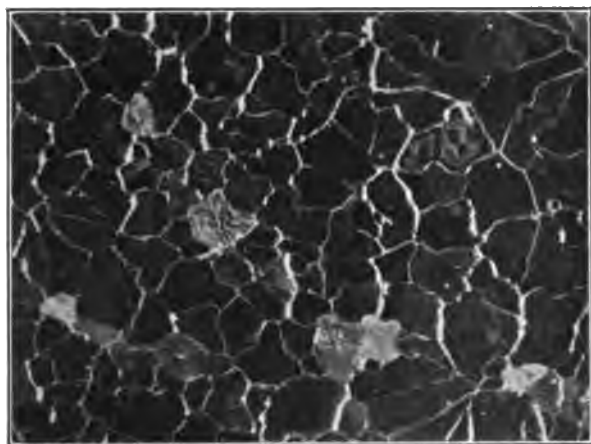


FIG. 8.—CAST SURFACE OF GOLD MORE QUICKLY COOLED. $\times 40$ DIAMETERS.

Each grain, then, is a crystal, imperfectly formed, it is true, because it has grown simultaneously from separate nuclei, symmetrical growth being stopped by final contact with neighboring crystals, after which only such

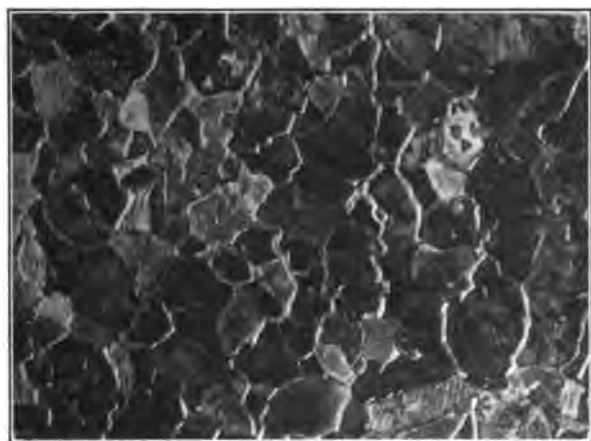


FIG. 9.—SAME AS FIG. 8, UNDER LIGHT AT A DIFFERENT ANGLE.

parts of the crystal have grown as would serve to fill up the liquid spaces remaining.

Other explanations¹⁰ have been advanced for the formation of crystal-

¹⁰ C. H. Desch: Solidification of Metals from the Liquid [State, *Journal of the Institute of Metals*, vol. xi, p. 57 (1914). (This contains complete references.)

line grains, but the assumption that growth proceeds simultaneously from separate nuclei receives the strongest support.

These true crystal grains are found also in metals which have been shaped by cold working. Fig. 10, showing a longitudinal section of swaged tungsten wire, illustrates this rather poorly, but it is given here to show the cold-worked structure of this material. The grains are seen to have been elongated.

Many experiments have been performed by many scientists, in attempting to determine what phenomena take place during cold working, by virtue of which the crystal grain would permit such distortion without the destruction of its general character. The studies of Rosenhain and Ewing,¹¹ who made a microscopic examination of polished metallic pieces during the actual straining operation, seem most satisfactorily to



FIG. 10.—LONGITUDINAL SECTION OF SWAGED TUNGSTEN WIRE. $\times 770$.

reveal the internal changes which take place. It was noticed during these tests that a number of fine dark lines appeared across the surface of each grain, being parallel in each but running in different directions in the different grains. This is illustrated in Fig. 11 which shows the surface of a distorted bead of cast gold. It was found that these lines or bands were, in reality, small steps produced by shearing at a corresponding number of internal surfaces. It was shown that each crystal grain behaves as does a pack of cards when bent or otherwise distorted, *i.e.*, by the sliding of very thin layers upon each other, so producing the "stepping" at the edges. These observed lines were called slip bands, and by their formation the grains are capable of a certain amount of deformation without breaking up.

¹¹ Ewing and Rosenhain: *Philosophical Transactions*, vol. cxviii, p. 355 (1900); and *Proceedings of the Royal Society*, vol. lxx, p. 85 (1899).

To explain how this distortion may probably take place within the mass of the crystal itself which, as a crystal, will not permit of flow, several theories¹² have been advanced, which, when combined and correlated, seem to clear up the matter very satisfactorily, for working purposes at least. It is well known that unequal pressure, when applied to a solid phase alone, lowers its melting point. It should be possible, then, by sufficiently increasing this pressure, to cause the material to melt at ordinary temperatures. Since both liquid and solid phases would at this point be in equilibrium, the molecular activity, or molecular freedom, of each phase would be the same. By equating values representing these molecular activities,¹³ and integrating, with the substitution of proper thermodynamic values, an equation has been evolved by Johnston and

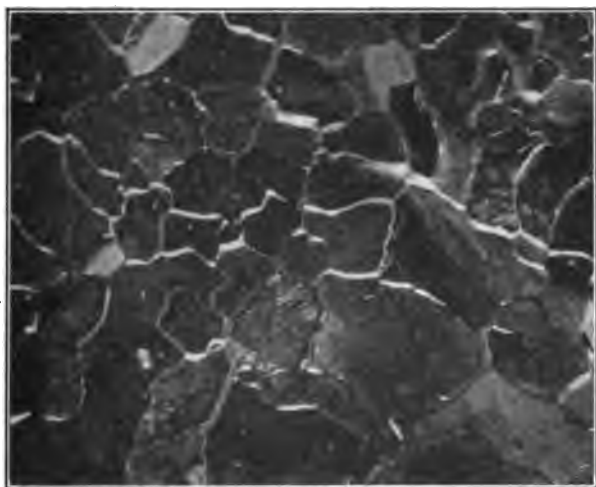


FIG. 11.—SURFACE OF A DISTORTED BEAD OF CAST GOLD. $\times 77$.

Adams, involving the density, heat of fusion, and normal melting temperature, from which may be calculated the pressures necessary to cause the different metals to melt at ordinary temperatures. This calculation resolves itself into the following final equation:

$$P = 95.1 \times Q \times D \times \text{Log} \frac{T_1}{T}$$

¹² G. T. Beilby: The Hard and Soft States in Metals, *Journal of the Institute of Metals*, vol. vi, p. 5 (1911); see also *Proceedings of the Royal Society*, vol. lxxix, p. 463 (1907), *Proceedings of the Faraday Society*, June, 1904, and *Philosophical Magazine*, vol. viii, p. 258 (1905). Johnston and Adams: High Pressure on the Physical and Chemical Behavior of Solids, *American Journal of Science*, vol. xxxv, p. 209 (1913).

¹³ Lewis: A New System of Thermodynamic Chemistry, *Zeitschrift für Physische Chemie*, vol. lxi, p. 129 (1907); also, *Proceedings of the American Academy of Science*, vol. xliii, p. 259 (1907).

in which P is the pressure, in atmospheres, necessary to cause fusion; Q is the heat of fusion of the compressed material; D , its density; T_1 , its normal melting temperature (Kelvin); and T , the temperature at which it is desired to cause the material to melt.

Johnston and Adams give the value of P for several metals, as follows: Sn, 2,200; Bi, 3,000; Cd, 3,300; Al, 5,100; Zn, 6,900; Ag, 14,000; Cu, 24,000; Pd, 31,000; Pt, 46,000 atmospheres.

The fact that layers of a solid, many hundreds of molecules in thickness, can really have this mobility of the liquid state conferred upon them by purely mechanical movement was first proved by Beilby,¹⁴ who found that a true skin is formed over the surface of a metal during polishing. This skin was found to be distinctly different from the crystalline material beneath it; it was harder, and even when formed on the surface of a crystal, of which the hardness varied in different directions, its hardness was the same in all directions. It was also found to be more readily attacked by solvents.

In polishing, and in any cold-working operation, it is conceivable that those particles which bear the brunt of the strain will be under a pressure corresponding to that at which they will melt at ordinary temperatures, and so will give way, flowing into areas of lesser pressure where resolidification takes place, leaving the load to be taken up by others in succession. According to this theory, during deformation of a metal, amorphous material is formed at all internal surfaces of shear (as indicated by the formation of slip bands), serving as a cementing material which is both harder and stronger than the crystalline material which it surrounds. This effect is illustrated in the increased tensile strength and stiffness of drawn wires.

Ordinarily this distortion is not carried to the stage at which the crystals are completely converted to the amorphous form, and hence the mass consists of one component in two phases, one of which is in a metastable condition. In the case of ordinarily undercooled liquids, the presence of the stable form tends to produce true equilibrium immediately, but in the case of the metals, the sluggishness of the molecules at this low temperature prevents this transformation. However, if the temperature be raised, the kinetic energy of the molecules rises, until, at a certain point, the fragments of crystals which remain are able to impress their orientation upon the molecules of the amorphous substance, and so slowly "absorb" it.

It will be evident that, above this temperature, the kinetic energy of the molecules is so great as to permit their returning at once to the crystalline form when displaced by mechanical distortion. The manipu-

¹⁴ G. T. Beilby: Surface Flow in Solids, *British Association Report*, Glasgow (1901); and The Surface Structure of Solids, *Journal of the Society of Chemical Industry*, vol. xxii, p. 1166 (1903).

lation of metals below this temperature is the familiar "cold-working operation; above this point, "hot-working." The crystalline state is that of maximum thermal stability and of minimum mechanical stability, while for the amorphous material the reverse is true.

According to the "amorphous cement" theory, it should follow, then, that a metal which has been cold-drawn to that stage which represents the formation of the maximum amount of amorphous material possible will be in the best condition to meet the imposed requirements of elasticity and stiffness.

Intercrystalline Boundaries.—The discussion has thus far included only such changes as may take place within the mass of the crystal itself, without a consideration of the phenomena by virtue of which the separate grains, or crystallites, are held together.

The internal strength of a single crystal, within which the molecules are arranged in a manner involving a minimum of potential energy, and hence of intra-molecular distance, may be ascribed to cohesion due to a mutual attraction of the closely packed molecules. Where two crystals meet, however, it seems improbable that the molecules of the different systems of orientation are close enough to one another to permit a degree of mutual attraction comparable to that existing between molecules of the same crystal. It might be supposed that the intercrystalline boundaries would be surfaces across which cohesion acted less strongly than it does within the mass of the crystal.

These intercrystalline boundaries would then be regarded as planes of weakness, when compared to the strength across any plane within the crystal. But this is not true, for the fracture of metals, in normal condition, always runs across the crystal grain¹⁵ rather than around the boundaries; and it is almost universally conceded, by those familiar with the microstructure of metals, that a fine-grained structure (in which these boundaries are numerous) is superior in reliability and strength to a coarse-grained one—which indicates that the intercrystalline boundaries are planes, not of weakness, but of strength. This has been ascribed by some to an interlocking of the irregular crystallites themselves, and by others to other causes; but a theory¹⁶ which receives strong support supposes the existence of a cementing material¹⁷ between the crystal grains. This theory assumes that when a metal crystallizes from the liquid state, many layers of molecules, lying at the planes where growing

¹⁵ W. Rosenhain: Deformation and Fracture in Iron and Steel, *Journal Iron and Steel Institute*, vol. lxx, Part II, p. 212 (1906).

¹⁶ G. D. Bengough: A Study of the Properties of Alloys at High Temperatures, *Journal Institute of Metals*, vol. vii, p. 9 (1912), and Discussion by Rosenhain, p. 176; also J. A. Sears: On the Longitudinal Impact of Metal Bars with Rounded Ends, *Transactions Cambridge Philosophical Society*, vol. xxi, p. 105 (1908).

¹⁷ Rosenhain and Ewen: Intercrystalline Cohesion in Metals, *Journal of the Institute of Metals*, vol. viii, p. 194 (1912) and vol. x, p. 119 (1913).

crystals meet, do not crystallize, perhaps because of the mutually neutralizing effect of adjacent crystal systems, and so solidify in the vitreous amorphous form. That this view is not untenable is shown in the case of amorphous silica, silicates, and similar substances¹⁸ which occur normally in the crystalline state also. The Phase Rule has, however, developed an excessive tendency to regard the stability which results from the crystalline state as the chief if not the only kind of stability which may exist, under ordinary conditions, in substances *normally* crystalline.

Such materials as glass, vitreous silica, etc., are harder and usually more brittle, but also much stronger, than the same substances in the crystalline form. Although these substances are essentially of the nature of liquids, they do not possess the mobility ordinarily associated with that state. Their viscosity at ordinary temperatures is very great, but they do possess the power of flowing to some extent, as shown by the bending of glass tubing under its own weight when placed at an angle for some time; and this property has been shown to exist even in vitreous silica.¹⁹

Substances of this type solidify to a completely amorphous mass because of the extreme sluggishness of their molecules at and below the freezing temperature, so that cooling may be comparatively slow without resultant formation of crystals.

In the case of metals, however, because of their extreme molten fluidity and long crystallization range below the freezing point, it has been found impossible to cool them from the molten state so quickly as to produce a completely amorphous solid. This has made it difficult to obtain direct experimental verification of the "amorphous cement" theory, as advanced in explanation of the strength of intercrystalline boundaries and the increased strength and hardness of cold-worked metals. It has been admittedly impossible, with the means at present available, to answer this question by any attempt to produce completely amorphous metals from the molten or crystalline states. However, by adopting a different method of attack, comparable to that employed in synthetic processes, the writer has, in a manner, overcome this difficulty—sufficiently so, at least to point out the remarkable way in which the physical properties of a simple metal may vary with its internal structure.

New Experimental Evidence in Support of the "Amorphous Cement" Theory (the Influence of Phase upon the Physical Properties of Gold)

According to the "amorphous cement" theory, then, the strength of the intercrystalline boundaries in metals is due to the presence of

¹⁸ Tamman: *Krystallisiren und Schmelzen*.

¹⁹ Kay: Thermal Hysteresis of Fused Silica, *Philosophical Magazine*, Series 6, vol. xx, p. 718 (October, 1910).

material which has not crystallized, and the increased strength of cold-worked metals is due to the formation of the amorphous phase during manipulation.

If the assumption that greater strength in metals is due to amorphous material present be sound, then a metallic mass known to contain the amorphous phase should possess properties differing from those of the metal when in the crystalline state.

The scheme adopted for testing this assumption involved the preparation of a series of gold briquets, made up of gold particles representing a different degree of subdivision for each briquet of the series, from coarsely crystalline to as near molecular proportions as could be obtained. This series consisted of four groups, each comprising several specimens for the different tests. No. 1 was made of pure cast gold, and represented a purely crystalline condition. No. 2 was made of fine gold filings, which represented a certain amount of amorphous material produced by extreme cold-working conditions. No. 3 was prepared from the precipitate formed by ferrous sulphate, and represented a very fine state of division.

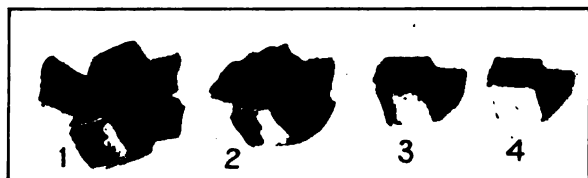


FIG. 12.—FLOW UNDER PRESSURE OF GOLD BRIQUETS OF DIFFERENT STRUCTURE.

No. 4 represented the ultimate limit of practicable subdivision. These briquets were prepared from a mud of colloidal gold, produced as follows: A deep red colloidal gold solution was prepared by reduction with 0.5 per cent. solution of hydroxylamine hydrochloride. This solution was carefully dialyzed to remove alkali salts, then evaporated to a "mud," and further dried on a water bath. It was found difficult to prepare a red solution containing more than 0.05 per cent. of gold, and even this, upon concentration, became greenish in color and resulted finally in a grayish-green sludge.

The size of gold particle, as determined for a red-gold colloidal solution, is of about .10 to 20 micro-microns. This size was no doubt increased, as indicated by a change in color, during evaporation. About 7 liters of this dilute solution was reduced for each briquet.

The apparatus used for briqueting this material was the same as that used for tungsten and molybdenum, and is described below.

The behavior of each of these series is strikingly shown in the photographs, Figs. 12 and 13. The different extent to which each would flow through the small openings between the different parts of the briquet

press is remarkable. Nos. 1, 2 and 3 were made under the same pressure, 9,700 kg. per square centimeter (about 135,000 lb. per square inch), and contain the same weight of gold. In the case of No. 4, the utmost pressure available, 19,300 kg. per square centimeter (about 265,000 lb. per square inch) did not cause any indication of flow. A similar striking difference is shown in the case of the Brinell hardness number of each of these series, using the large ball and a pressure of 500 kg. No. 1 was too soft to bear this weight, so the hardness number was taken from a bar of rolled and annealed gold and found to be 23.80. The hardness number for No. 2 was 38.10; for No. 3, 53.40; and for No. 4, 94.70. These depressions are shown comparatively in Fig. 13.

Other striking phenomena were noted in connection with these series, but their behavior under pressure will serve to point out the fact that much remains to be discovered regarding the physical properties of simple metals, as well as what may be expected should this knowledge lead to a control of phase.

The important feature of these results, however, is the apparent support they give to the amorphous cement theory of Beilby, Rosenhain, and others, above. Since these results have shown that characteristics

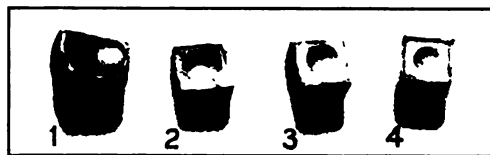


FIG. 13.—COMPARATIVE HARDNESS OF BRIQUETS IN FIG. 12.

of an amorphous metal are great hardness and resistance to deformation, nothing could more nearly explain the increased strength of the cold-worked metals than a theory assuming the formation of amorphous material during manipulation.

3. Practical Deductions

With the application of these crystallographic considerations, the probable cause for the difficulties encountered in the production of ductile tungsten, and to a greater extent, of ductile tungsten alloys, seemed apparent. Careful microscopical examination of many specimens of drawn wire submitted to this laboratory revealed in all cases that brittleness is accompanied by a difference in crystal structure. Fig. 10, already described, showed a longitudinal section of drawn tungsten wire, which was very tough and flexible. Fig. 14, a longitudinal section of a similar wire, which was exceedingly brittle, clearly shows the discontinuity of the drawn, fiber-like structure.

In discussing the internal structure of metals it was pointed out that a sufficiently high temperature would permit the complete crystallization

of amorphous metal. The larger crystals of this mass may impress their orientation upon the molecules of smaller crystals, thus causing crystal growth; and mechanical distortion, above a certain temperature, will not greatly reduce this grain size.

In devising a method for producing tungsten and its alloys in a non-brittle form, these points were regarded as fundamental. The brittleness of the examined specimens of drawn tungsten was regarded as due to an insufficient breaking up of its crystalline structure. Hence, in producing this material, or its alloys, in ductile form, it would be necessary to eliminate excessive crystalline material; and in order to accomplish this, the following means suggested themselves, based upon deductions drawn from earlier conclusions regarding the behavior of worked metals in general.

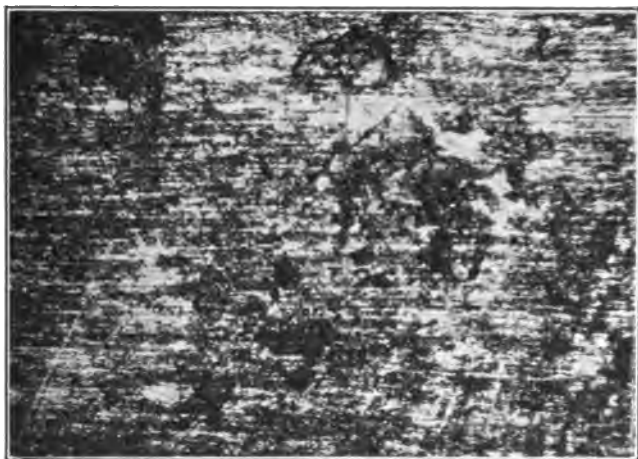


FIG. 14.—LONGITUDINAL SECTION OF BRITTLE TUNGSTEN WIRE. $\times 743$.

1. In order to produce a practical minimum of crystal size in the starting ingot, the treating temperature must be as low as possible.
2. In order to decrease the amount of crystalline material, cold-working conditions must prevail.
3. In order to introduce sufficient distortion, with its accompanying formation of the maximum amount of amorphous material, resulting in greater ductility in the larger sized wires, drawing should be commenced at an earlier stage in the reduction of the briquet, because this operation is much more effective in breaking up a crystalline structure than is swaging.
4. For obvious reasons, the above operation should be controlled by microscopical examination. Internal structure, not temperature, is the treating criterion.

That these premises are fundamentally sound is shown by the following results of experimental work.

V. EXPERIMENTS

The experimental work, which involved the application of the above considerations, and the results of which are outlined in this paper, included investigations of the following series: pure tungsten, tungsten containing 0.75 per cent. ThO_2 , pure molybdenum, tungsten-gold, tungsten-palladium, tungsten-molybdenum.

The method adopted comprised the compression of amorphous powder to briquet form; the heat treatment of the compressed material; and the forging of the resulting ingot.

The tungsten and molybdenum used in these experiments were obtained through the courtesy of the Cleveland branch of the General Electric Co., and were the chemically pure material used for the production of drawn wires.

The tungsten was of two batches, one containing 0.75 per cent. of ThO_2 which is the material largely used for lamp filaments, while the

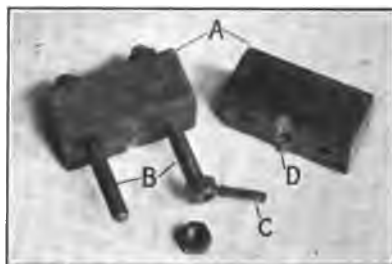


FIG. 15.—BRIQUET MOLD.

other was tungsten in which no trace of impurities could be detected. The gold and palladium were purified in this laboratory by several re-precipitations. The preparation of the amorphous material, from which briquets were made, will be described in connection with the different series.

1. *Compressing Powder to Briquet Form.*

The mold used in producing the raw briquet is shown in Fig. 15. This consists of two halves of hardened tool steel, held together by strong bolts. Into each half is milled a polished V-shaped groove, so that when the two parts are bolted together a square chamber is produced. Into one end of this chamber the plug *D* is inserted, and after filling the chamber with the material to be compressed, the plunger *C*, is forced into place. This plunger is steel containing tungsten, 16.5; chromium, 2; and carbon, 0.6 per cent., which, after heat treatment, has been tested under pressures exceeding 300,000 lb. per square inch. The face of the plunger was highly polished so that one surface of the raw briquet was available for micro-examination.

The design shown in Fig. 16 was a failure, since the adhesion between the briquet and the semi-cylindrical surface of the mold was greater

than the cohesion across the section within the briquet itself, causing it to split while being removed.

Compression was obtained in a Brinell hardness-testing machine, the



FIG. 16.—UNSATISFACTORY BRIQUET MOLD.

ball, with its socket, being replaced by a special head. The press A was placed upon the adjustable table as shown in Fig. 17, so that the plunger came axially under the piston of the machine. This machine gave very

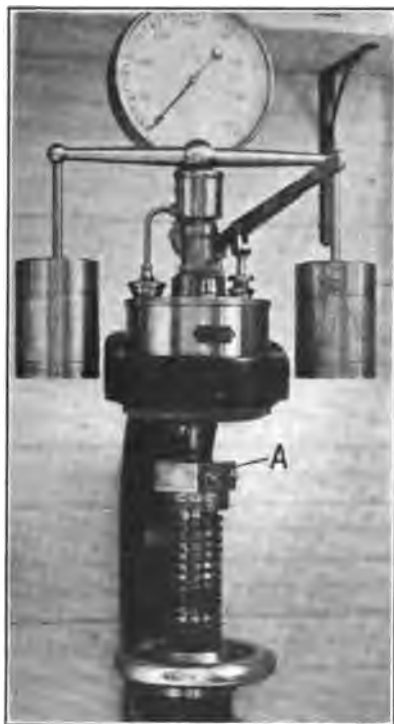


FIG. 17.—MACHINE FOR COMPRESSION OF BRIQUETS.

sensitive and accurate control up to a total load of 4,000 kg. Various total loads, when exerted upon the piston of 0.207024 sq. cm. in cross-section, are given in comparative values as follows:

Total Pressure, Kilograms	Kilograms per Square Centimeter	Atmospheres	Pounds per Square Inch
500	2,412.7	2,335.5	33,561
1,000	4,825.5	4,671.0	67,122
2,000	9,651.0	9,342.0	134,244
3,000	14,476.5	14,013.0	201,366
3,750	18,095.5	17,516.0	251,707
4,000	19,302.0	18,684.0	268,488

2. Heat Treatment of the Compressed Material

Quartz-Tube Furnace.—Fig. 18 shows a quartz-tube furnace which was used for the reduction of tungsten and molybdenum, and other oxides

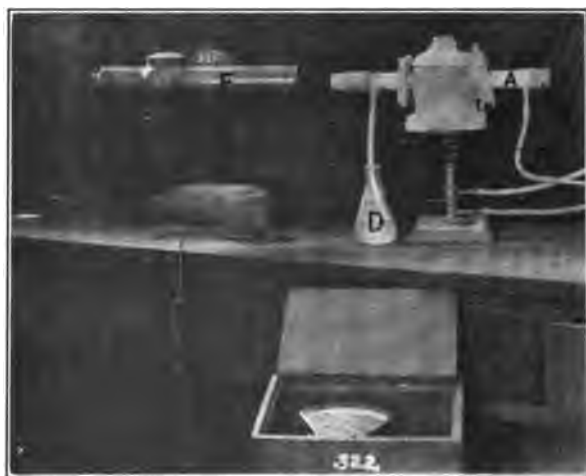


FIG. 18.—QUARTZ-TUBE FURNACE.

and salts; also for various purposes where heating in controlled atmosphere was desired but where very high temperatures were not necessary. In this, temperatures up to 1,225°C. are easily and quickly reached. The fused quartz tube, *A*, has a glass window at one end, and at the feed end the plug, *B*, which can be removed for the insertion of the boat containing the material to be treated. Hydrogen, or other gas, enters through tube *C* and escapes through the bubbler *D*. A Meker blast lamp, *E*, was used for heating. The method of sighting with optical pyrometer *F* is also shown.

Gran-Annular Electric Furnace.—For temperatures up to 1,800°C. this furnace far surpasses any other type which the writer has used in experimental work of this kind.

Figs. 1 and 2 showed this furnace in detail. Fig. 19 shows a setup of the furnace, *A*, in series with cast-grid rheostat, ammeter, and switchboard. An optical pyrometer, *B*, points directly into the heating chamber of the furnace.

Electrical Resistance Furnace (Resistance of Charge).—The above furnace equipment, however, did not provide for the treatment of tungsten and molybdenum briquets, or those containing high percentages of these metals. Moreover, it provided no means for forging these ingots. This operation had to be carried out at high temperatures and in an inert atmosphere. Fig. 20 gives a vertical section of a furnace so designed as to



FIG. 19.—GRAN-ANNULAR ELECTRIC FURNACE WITH CONNECTED APPARATUS.

furnish temperatures ranging from that of the furnace room up to, and above 3,000°C., depending upon the material of the briquet, and also to serve as a forge for working the material into a solid mass while at a high temperature. This consists essentially of two adjustable electrode bars, *A* and *B*, fitted with fused tungsten contact surfaces, *C*, between which the briquet, *D*, is placed. Current is passed through the briquet which, by its own resistance and that between the contact surfaces, may be heated to any temperature up to its melting point. An inclosed heating chamber, *E*, is formed by dropping the sliding cup, *F*, into a close-fitting recess made for it at the base. Gas is passed through the tube, *G*, into this chamber,

from which it escapes through the outlet shown. This opening serves also as a peep hole for the optical pyrometer.

In operating, the bar, *A*, with its electrode and housing, *F*, is raised through its guides, *H* and *I*. The briquet, *D*, is then placed in position on the lower electrode, *C*, after which the upper electrode is lowered to contact. The inverted cup, *F*, is then lowered into place, forming the chamber *E*, through which is passed an inert gas, swiftly at first, to expel the air, and

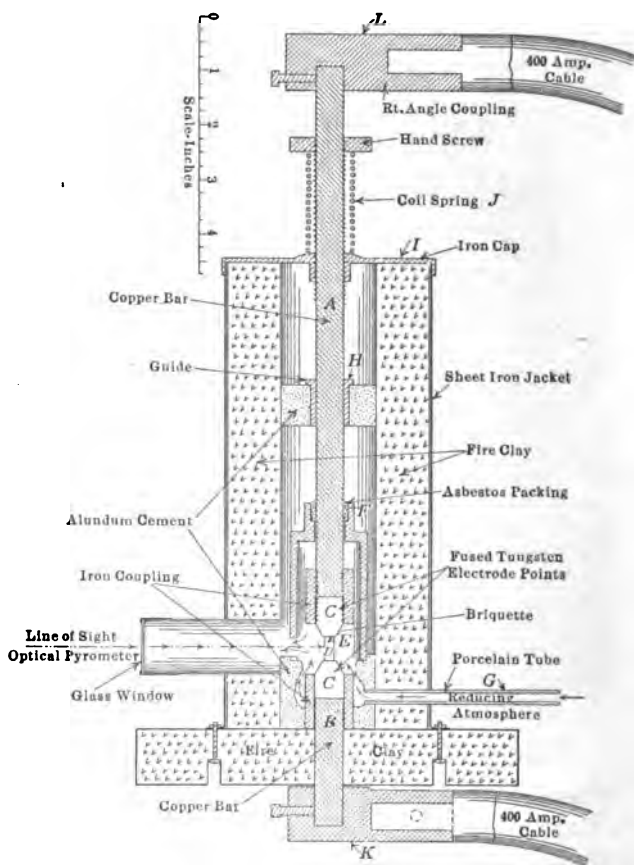


FIG. 20.—ELECTRICAL RESISTANCE FURNACE.

then slowly, as indicated by a bubbler in the purification train. To avoid the fracturing of very fragile briquets, and to prevent a resulting short circuit, the coil spring, *J*, with adjustable collar, is placed so as to support the excess weight of bar and cable.

Fig. 21 is a photograph of this furnace with its door removed so as to show the different parts in place for heating. Fig. 22 shows the complete setup: *A*, being the furnace; *B*, a hydrogen generator; and *C*, an



FIG. 21.—ELECTRICAL RESISTANCE FURNACE.



FIG. 22.—ELECTRICAL RESISTANCE FURNACE, WITH CONNECTING APPARATUS.

optical pyrometer in position for reading the briquet temperature, which is indicated on the scale contained in the case *D*.

Forging

This furnace was designed to be used as a forge, and at the same time to maintain the temperature at which the material of the briquets could be shaped. During this operation the lower bar was firmly supported, while blows were struck with a hammer upon the upper electrode. The increasing of the cross-section and the welding of the material would reduce the resistance considerably, resulting in a lowering of the temperature, unless an increased current were used; but, by pyrometric observation and current regulation during the forging operation, the temperature could be kept fairly constant. The peep hole permitted direct observation of the briquet at all times.

Temperature Measurement.—The conditions under which high temperatures were to be maintained did not permit the consideration of any thermometer or thermocouple method of pyrometry. The practical methods available were those based on the various radiation formulæ, or upon optical observation. A review of this branch of pyrometry revealed an abundance of literature,²⁰ upon the governing principles of these methods, and their practical application. Of the various types of instrument of this class available, that one was chosen as most practical under the prescribed conditions which involves a direct telescopic comparison of the luminous radiation of the hot body with that of a standard source. There are two makes of this type; one, the Holborn-Kurlbaum,²¹ and another, of almost identical construction, the newer type of Morse Thermogage.²² The working principle is the same in both. A current is passed through the filament of a lamp, which first becomes red and then, with increasing current, successively orange, yellow and white. By interposing this filament between the eye and the hot body to be measured, the current through the lamp may be adjusted until the middle of the filament is of the same color and brightness as the object. At this point the part of the filament used for comparison will disappear against the bright background. The current through the lamp, interpreted in terms of temperature, gives the temperature of the hot body. An absolute match of both color and brightness cannot be made, unless monochromatic light is used, because of the fact that the lamp filament and the viewed body do not radiate similarly. This comparison is facilitated by a

²⁰ G. K. Burgess: *Measurement of High Temperatures* (A complete treatise and bibliography). See also C. E. Mendenhall and Forsythe, *Physical Review*, vol. iv, p. 163 (1896-7).

²¹ Made by Siemens & Halske Aktiengesellschaft, 94 Markgrafenstrasse, Berlin.

²² Made by Morse Thermogage Co., Trumansburg, N. Y.

suitable arrangement of lenses and lamp, as in the instrument of Morse, which was the one used in this work. This is shown diagrammatically in Fig. 23. A low-voltage incandescent lamp, *L*, with M-shaped filament is mounted in the focal plane of the objective, *O*, and the eyepiece, *E*, of a telescope is provided with means of focusing. The lamp circuit is shown connected through a battery, a rheostat, and a milliammeter. Fig. 24 shows this instrument with positions of the rheostat, *A*, and lamp, *B*. Fig. 25 shows a portable case containing battery, milliammeter, and pyrometer.

Temperatures were determined by focusing the instrument upon the briquet so as to bring its image into the plane of the lamp filament, then adjusting the current, by means of the rheostat, until the tip of the filament disappeared against the image of the briquet, when the temperature was read directly from the milliammeter scale (previously calibrated by checking current readings against known temperatures).

Below 1,200°C. readings were made directly, while above this tem-

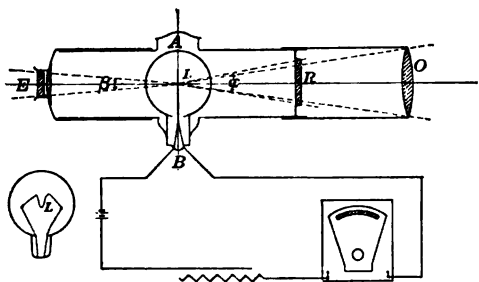


FIG. 23.—DIAGRAM OF MORSE THERMOGAGE.

perature a neutral-colored absorbing "Rauch" glass (*R*, Fig. 23) was interposed between the lamp and the objective, in order to prevent the blinding effect of too great brightness, and also, to avoid overheating the standard filament. A new calibration was necessary when this absorbing screen was used. For temperatures above 1,900°C. it was found necessary to cut down the amount of light entering the pyrometer by means of a diaphragm, with the use of which temperatures up to that of the arc could be read.

The instrument was carefully standardized in all three ranges. The first, in which no absorbing glass was used, was carefully checked against an accurate thermocouple. The second range, using an absorbing light screen, was checked up to 1,375°C. against a thermocouple, and at higher temperatures by the freezing points of palladium and platinum. The Gran-annular electric furnace was used in this standardizing work up to, and including, the melting temperature of platinum, and for this purpose approximated very closely the desired black-body conditions. These standardizations gave current-temperature curves up to nearly 1,800°C.,

which could be continued by interpolation, with a fair degree of accuracy, up to $2,000^{\circ}\text{C}$.

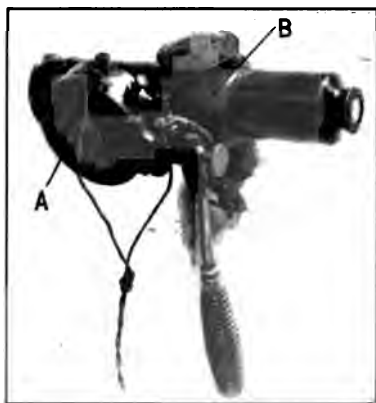


FIG. 24.—MORSE THERMOGAGE, WITH RHEOSTAT AND LAMP.

A third curve was formed by the melting points of platinum, iridium ($2,200^{\circ}\text{C}$.) molybdenum ($2,500^{\circ}\text{C}$.) and tungsten ($3,000^{\circ}\text{C}$.). These latter points were obtained by sighting on electrically heated strips of the



FIG. 25.—PORTABLE CASE, CONTAINING BATTERY, PYROMETER, ETC.

metals formed into narrow wedges, as recommended by Mendenhall,²² which produces a deep cavity giving very near block-body conditions.

²² Burgess and Le Chatelier: *Measurement of High Temperatures*, 3d edition, p. 334 (1912).

The tests on tungsten and molybdenum were made by grinding very thin a portion of a piece of wire, which, on account of the resulting small cross-section, heated very readily. These tungsten and molybdenum strips were arranged as lamp filaments in a glass tube, *in vacuo*. Readings were also made on fusing briquets of these metals, when heated, under operating conditions in the above-described resistance furnace.

A final reading was made on the positive crater of the arc, accepting for this a temperature of 3,600°C. It has been shown that an absolute match of both color and brightness cannot be made, unless monochromatic light is used, or unless the lamp filament and object radiate similarly, which is seldom the case. All observations were, therefore, taken through red screens, which, with a radiation characteristic of about 2,000°C., gave a very narrow band with maximum of transmission at $\lambda = 0.659\mu$.

In all readings care was taken to maintain axial symmetry and constant distance from the hot body to the objective, thus insuring a sufficiently large background, and also constant angles at ∞ and ζ .

The accuracy of this method of pyrometry has been fully proved and is such that between 1,000°C. and 2,000°C. the error of reading should not exceed 10°C. Above this range the error was found to be somewhat greater, but not so great as to interfere in any way with a duplication of desired conditions.

Metallography

Grinding and Polishing.—After heat treatment the ingots were made ready for microscopical examination. The preparation of specimens of this small size (about 0.5 by 0.5 by 1.0) cm. was rather a delicate operation, but the final surfaces secured were all that could be desired. Excellent apparatus was available for this purpose, while the three grades of alundum, F, XF, and 65F, were found to be sufficient for polishing all samples. Care was taken during these operations not to introduce surface conditions which would mask important internal phenomena.

Etching.—For pure tungsten and molybdenum or their high-percentage alloys, the most satisfactory etching reagent was found to be concentrated hydrogen peroxide. It is necessary, however, to have this boiling while in contact with the specimen, otherwise it is quite inert. For the alloys containing a precious metal also, either hydrogen peroxide or aqua regia was used, depending upon whether the tungsten and molybdenum, or the precious metal was to be attacked.

Microscopy.—For microscopical examination, there was available, from both Carl Zeiss and E. Leitz, a very complete equipment, the detailed description of which is not deemed necessary.

Results

In order to test the soundness of the premises upon which the experimental work was based, it was necessary that experiments should first

be directed toward the production of workable masses of pure tungsten and molybdenum themselves, for, should this not be possible, it would be useless to extend the work to their alloys; and, conversely, should these pure metals yield to such systematic treatment, as based upon conclusions drawn from earlier study of the probable theoretical conditions involved, then the application of similar methods should meet with success in the treatment of their alloys.

It must be pointed out here that no treatment would overcome defects in these alloys due to chemical interactions, but that if, for instance, brittleness, as in the case of tungsten and molybdenum, is due to a purely physical condition, then a method based upon the above considerations should meet with success.

There are two fundamental critical points to be determined: First, the proper ingot-forming temperature; and second, the lowest possible

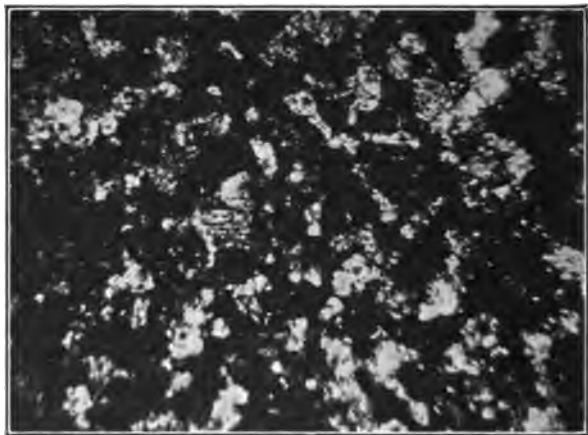


FIG. 26.—RESULT OF LONG, RELATIVELY LOW HEATING.

temperature at which this ingot will yield to mechanical working. This latter point must obviously be below the hot-working range.

With the above considerations in mind, and in accordance with the principles laid down in the preceding pages, the experimental work has been carried out, with results as shown briefly under the following separate heads: (1) Pure tungsten; (2) pure tungsten plus 0.75 per cent. ThO_2 ; (3) pure molybdenum; (4) tungsten-gold series; (5) tungsten-palladium series; (6) tungsten-molybdenum series.

Pure Tungsten.—The pure tungsten briquets of this series were made under a pressure of 14,500 kg. per square centimeter (about 200,000 lb. per square inch). The specific gravity of the resulting briquet was 13, taken as an average of nine specimens. This was determined from micrometer measurements of volume, and weight of the specimen, and represents about 30 per cent. of void in the briquet.

From numerous results, the following are taken as typical of the changes which take place upon treating the amorphous briquets at various temperatures:

TABLE III.—*Results of Heat Treatment of Briquets*

Number of Specimen	Treating Temperatures	Time of Heating	Remarks
	Deg. C.	Hr. Min.	
5	500	1 0	Fairly coherent. Partly sintered. Could not be polished.
6	600	1 0	Harder and stronger than No. 5. Could be poorly polished.
8	800	1 0	Hard and metallic. Filed with difficulty. Polished readily.
10	1000	1 0	Very hard and metallic. Polished to beautiful surface.
12	1200	1 0	Appeared microscopically as solid metal. Very hard and brittle.
14	1400	1 0	As No. 12. No crystal formation.
16	1600	1 0	An apparent internal change, which could not, however, be called crystalline.
18	1800	1 0	Showed changes indicating crystal formation, but with no definite outlines. (Gran-Annular furnace.)
22	2200	0 10	Appeared about as No. 18. (Briquet resistance furnace.)
26	2600	0 10	Very finely crystalline with many intercrystalline boundaries not clearly defined—indicating presence of amorphous material.
27	2700	0 5	More coarsely crystalline than No. 26 with crystals in center larger than at sides.
28	2800	0 1	About as coarsely crystalline at surface as No. 28, but area in center indicated a complete liquid fusion.

The extent and type of crystallization were found to depend upon the time-temperature factor. Prolonged heating at relatively low temperatures gave fewer and larger crystals, and crystals of different sizes (Fig. 26), while a flashing at higher temperatures (specimen No. 26) gave a very fine and uniform crystalline structure (Fig. 27), which was much to be preferred. In specimen No. 28 (Figs. 28 and 29), the heat of radiation from the sides of the briquet was so great that while the surface-temperature reading was only 2,800°C., the molten interior indicated a temperature above the melting point of tungsten (3,000°C.) at the center. Photomicrographs, Figs. 30 to 32, show the transition from the compressed powder to a solid, metallic, apparently non-crystalline mass, and finally to a solid, crystalline ingot.

The next step was to determine which of these stages of crystallization gave the best forging conditions. This was done by treating a series of each of these numbers at varying temperatures. It was found that,

within certain limits, the lower the sintering temperature, the higher was the initial temperature necessary for forging.

No. 5 could be forged at about 2,400°C.; No. 10 at about 2,200°C.; No.

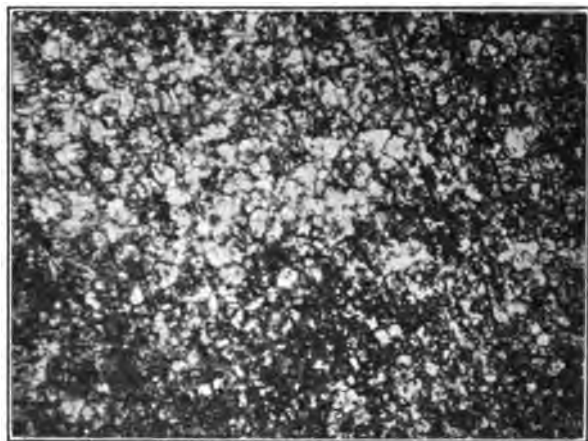


FIG. 27.—RESULT OF QUICK HIGH HEATING.

18 at about 2,000°C.; No. 22, at about 1,700°C.; No. 26, at 1,275°C.; No. 27, at 1,350°C.; No. 28, at 1,375°C. These figures, of course, represent the initial welding temperature of the more or less powdery mass, the

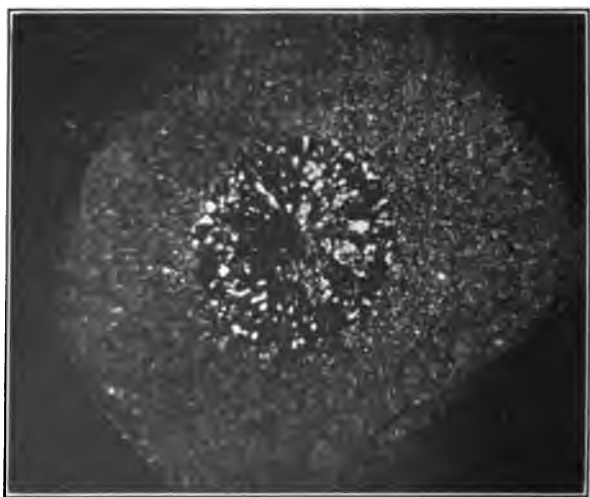


FIG. 28.—MOLTEN INTERIOR, SPECIMEN 28.

necessary temperature falling at once when this welding had begun; they give no definite information other than indicating the necessity for at least partial crystallization in the treated ingot. Below these high

temperatures the specimens were brittle in the first stages of forging, and if these temperatures were maintained after forging began, there resulted a marked crystal growth, instead of the desired reduction.

These series were represented by 92 forgings, or attempted forgings,

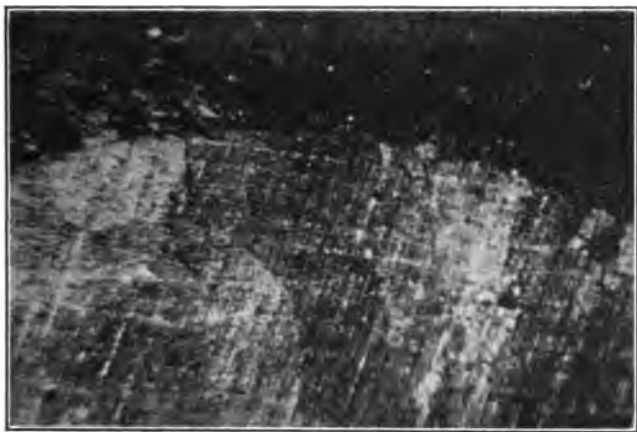


FIG. 29.—SURFACE, SPECIMEN 28.

and the results gave a curve with a minimum forging temperature corresponding to a certain crystal size. This is represented approximately by No. 26 (micrograph Fig. 27). A higher temperature for a shorter time would give nearly the same conditions, or a lower temperature for a longer



FIG. 30.—FIRST STAGE, COMPRESSED POWDER.

period. It is thus seen that it would be useless to prescribe a definite treating temperature, or even one for forging, because the temperatures recorded were observed under rather restricted conditions, and in apparatus giving the worst possible conditions as to radiation, uniform heating, etc., due to the small scale on which the operations were carried out.

The results, however, showed how dependent are the physical properties of wrought tungsten upon the heat treatment and the crystallographic control exercised during the various stages of its manufacture.

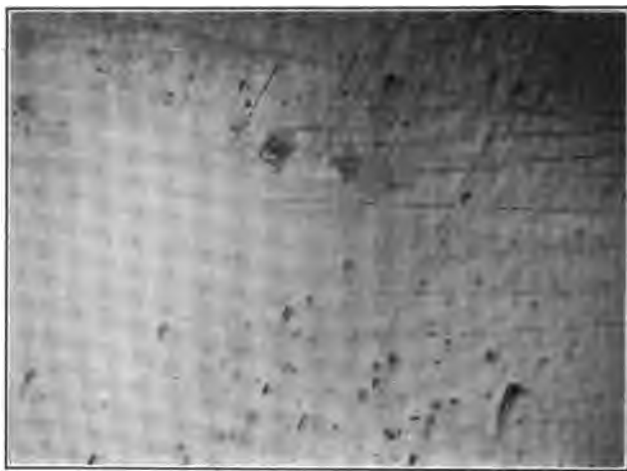


FIG. 31.—SECOND STAGE, NONCRYSTALLINE MASS.

The best specimens of this series were hammered out to a thickness of less than 0.5 mm. and, although it was necessary to increase greatly the heating current with increased cross-section of the ingot, the contact



FIG. 32.—THIRD STAGE, SOLID CRYSTALLINE INGOT.

resistance was enough to maintain a sufficiently high temperature. As the electrodes approached each other it became increasingly difficult to make accurate temperature measurements, since only the irregular outside edge of the flattened ingot could be sighted upon with the pyrometer.

No means were available with which to draw this material, so that its behavior during that operation, and the influence of the above-described control can only be assumed.

Tungsten with 0.75 per cent. Thorium Oxide.—An exactly similar series of experiments revealed only one point in which tungsten and thorium oxide reacted differently from the pure tungsten; but that point is of the greatest importance.

Sintering progressed with the gradual appearance of crystallites in a manner so closely paralleling the preceding case that the presentation of another almost identical set of figures and micrographs is thought unnecessary. But with respect to the crystal growth at higher temperatures, there was a marked difference. The presence of this small amount of thorium oxide was found largely to prevent a coarsely crystalline formation in highly overheated specimens; and even at stages just below the actual melting point the structure was very uniform. The explanation for this marked influence of thorium oxide is probably as follows:

Crystallization of this powdery, amorphous mass no doubt begins, as in the case of crystallization from the liquid state, at numerous centers, and proceeds, as in that state also, by the impressed orientation and resulting "absorption" of adjacent molecules of amorphous material, accompanied by an expulsion of foreign matter outward until, when neighboring crystals meet, the intercrystalline boundaries consist of thin sheets of this foreign matter. In most known cases the presence of this foreign material produces a weakness in the entire mass, but in the case of tungsten, the thorium oxide seems to be in such form as not only to prevent crystal growth, by its presence among the molecules of tungsten and later, between adjacent crystallites, but to maintain the strength of the entire mass. This may be a phenomenon similar to that which may be caused by iron oxide when two pieces of iron are welded together.²³ It has been noticed, in this case, that no matter how great the crystal growth occurring within either of the welded parts, no growth will extend across the weld; and this weld is usually the strongest section of the specimen.

Several experiments were performed, in which the thorium oxide was replaced by other refractory oxides. Magnesium oxide, cerium oxide, and other refractory earths, produced a similar retarding action upon crystallization, but the effect was not so marked. Although the experiments on other oxides than thoria were not sufficiently extended to locate these materials on a scale of their comparative effects in retarding the crystallization of tungsten and molybdenum, their value, for this

²³ It might be of interest to determine the effect of iron oxide upon the crystallization of iron, and upon its physical properties, by adopting a method similar to that here used for tungsten.

purpose, seems to be a function of the temperatures at which they begin to soften.

The explanation of the accumulation of thorium oxide at the inter-crystalline boundaries was first given by Zay Jeffries;²⁴ and this phenomenon has been observed so often, during studies in connection with these experiments, that the logic of the explanation seems proved. Nothing is known regarding the physical characteristics of this rare earth when segregated under these conditions, but there is no reason for believing that it may not be in a vitreous, amorphous form, and hence, of great strength.

The influence of thorium oxide during forging was not so marked, but this is due to the fact that these experiments were carried out only to such extent as to prove that this material could be forged under the above condition. Indirectly, however, the effect of thorium oxide was present, in

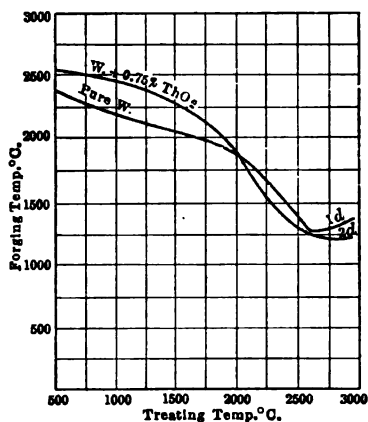


FIG. 33.—TREATING AND FORGING TEMPERATURES FOR PURE TUNGSTEN AND TUNGSTEN CONTAINING 0.75 PER CENT. OF THORIUM OXIDE.

that those ingots formed at the higher temperatures, and in which excessive crystallization had been prevented by its presence, could be forged at the lower temperature, the curve not showing an upturned branch at the high-treating temperature end, as had that for pure tungsten (see Fig. 33).

Pure Molybdenum.—The material used in this series, obtained from the General Electric Co., was the chemically pure powdered metal used in the manufacture of drawn wire.

The raw briquets of molybdenum were made under a pressure of 14,500 kg. per square centimeter and gave an average specific gravity of 7.80. Table IV shows the change in internal structure of briquet with increasing temperatures.

²⁴ Private communication, 1914.

TABLE IV.—*Structural Changes in Briquets of Molybdenum*

Specimen Number	Temperature, Deg. C.	Time, Hr. Min.		Remarks
4	400	1	0	Slightly harder and more coherent than raw briquet.
8	800	1	0	Hard enough to be poorly polished.
12	1,200	1	0	Very hard and brittle, but of considerable compressive strength. Polished as solid metal.
16	1,600	1	0	Stronger and more metallic than No. 12.
20	2,000	1	0	Structure changed. Showed patches which microscopically behaved as solid crystalline material, but with no definite boundaries.
22	2,200	1	0	Coarsely crystalline, but with grains of very irregular size.
24c	2,400	0	10	Coarsely crystalline, grains of more regular size than 22.
23g	2,300	0	1	Very finely crystalline. Ingot quite homogeneous throughout, best structure of series.

The photomicrographs of this series could not be distinguished, in representing the transition from compressed amorphous powder to crystalline ingot, from those for pure tungsten, and hence are not given.

Specimen No. 23g, was of minimum grain size consistent with maximum crystalline formation. In this, as in the preceding series, it seemed necessary to reach a certain size of grain in order to produce an ingot free from patches of incompact amorphous material, the presence of which made it necessary to use a very high temperature at the beginning of the forging operation. On the other hand, too high a treating temperature produced a coarsely crystalline structure, with its inherent brittleness, which also could not be overcome, except at high temperatures.

The forging curve for pure molybdenum showed a minimum corresponding to No. 23g, with higher temperatures on either side, as in the case of pure tungsten.

The results of experiments on tungsten and molybdenum have given excellent support to the theory upon which the work was based, and furnished details of critical points which promise to be paralleled, and should be searched for, in any series of their alloys.

Having established the soundness of the principles involved when applied to these pure metals, the next step was to apply them in connection with their alloys.

It must be again pointed out that this method did not have as its object the production of true alloys, in the sense of those which solidify from the liquid state, but to avoid such consequences as might result from

that condition, by welding together, at a temperature below the melting point, particles of molecular proportions, which should be as evenly distributed and uniformly mixed as are the molecules in a true solid solution.

Tungsten-Gold Series.—To insure an intimate mixture of gold and tungsten the chemically pure tungsten metal was first oxidized and then saturated in an evaporating dish with an acid gold chloride solution containing the desired percentage of gold. This was carefully evaporated and dried, with constant stirring to prevent segregation; then transferred to an alundum boat and placed in the quartz-tube furnace for reduction under pure hydrogen. This material, when compressed into briquets, under 19,300 kg. per square centimeter (about 265,000 lb. per

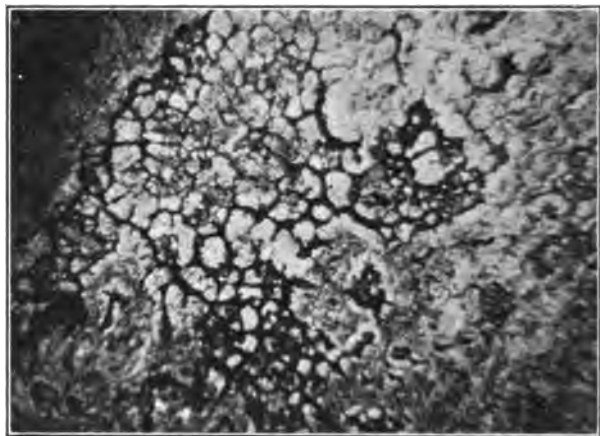


FIG. 34.—TUNGSTEN-GOLD MIXTURE, CONTAINING MORE THAN 1.4 PER CENT. OF GOLD. DARK AREAS, GOLD. ETCHED WITH AQUA REGIA.

square inch) pressure and examined under 2,500 diameters, did not show the slightest non-uniformity.

This series was investigated through a range from 0.1 to 20 per cent. of gold; but to give in detail the numerous negative results obtained would make this paper too cumbersome. A brief description of the behavior of these combinations in several proportions will suffice for the series.

No combination in this series could be forged at any temperature; even 0.1 (which was the smallest percentage of gold used) seemed to be sufficient to prevent working of the mass.

Below 1.4 per cent. the gold could not be detected as segregated at any stage of the treatment. This amount, however, was sufficient to cause coarse crystallization at about 2,500°C. Analysis showed the rather remarkable fact that very little of this gold was lost, even in those specimens which were treated to a temperature of 2,600°C., which is far above the accepted boiling point of gold.

The extreme fragility of these specimens at very high temperature prevented even an approximate determination of the melting point.

Above 1.4 per cent. of gold, a segregation resulted in those specimens treated between about 1,200°C. and 2,200°C. (see Fig. 34). Below about 1,200°C. no change could be detected microscopically, while above 2,200°C. a homogeneous crystalline mass resulted. An interesting phenomenon is noted in connection with temperatures above this latter point. In the case of specimens containing 5, 7, and 10 per cent. respectively of gold, after being heated to a temperature near 2,600°C. the gold content was found to be partly volatilized, and condensed in visible particles on the chamber walls and on the electrodes. In each case there remained a coarsely crystalline mass containing from 4 to 5 per cent. of gold.

No attempt was made to learn the nature of this material or to determine its properties, other than to prove its worthlessness from a mechanical point of view.²⁵

Tungsten-Palladium.—A similar investigation of the tungsten-palladium series, but involving only 0.0 to 4.5 per cent. of palladium, gave results very similar to those of the preceding series. Below the melting point of palladium, no change could be detected. Above this point the mass became crystalline and possessed an inherent brittleness which could not be overcome. This was no doubt due to the fact that some reaction had taken place. Whether a compound or solution was formed is not known, but that combination of some kind had occurred, was attested by the fact that the mass appeared homogeneous at all stages of the treatment under the highest magnification.

These ingots of tungsten containing palladium were not found to be malleable, when worked under the above-described conditions.

Tungsten-Molybdenum Series.—The tungsten-molybdenum series was investigated in various proportions of the components, taken at 10

²⁵ It would be interesting to try this method upon combinations of metals such as nickel and palladium, copper and gold, or a similar series, whose components were of nearly equal melting point; for the failure to produce results in the case of gold and palladium with tungsten may be ascribed to a too great difference between melting points of the components. Near the lower range, the tungsten is not sufficiently plastic to permit a welding between the particles; while at higher temperatures, fusion conditions are present.

Even the pure metals, as gold, copper, iron, etc., might, if prepared in this way, possess properties far superior to those of the cast and worked metals, for in this manner the mass may be made to consist more completely of amorphous material, with its attendant characteristics, as shown in the results of experiments on pure gold, as described above. Even though an entirely amorphous condition could not be maintained throughout the operation, at least a much more finely crystalline ingot might result than from any available fusion method.

Experiments have been started with the view of testing this out, but have not progressed to that stage which would permit a definite statement regarding this possibility.

per cent. intervals through the range from pure tungsten to pure molybdenum. The alloys represented by each of these intervals were found to be susceptible to certain conditions under which they could be wrought. The briquets were made under a pressure of 18,000 kg. per square centimeter and were then treated through a range of temperature in a manner similar to that employed in the previous series. Figures for the treating and forging temperatures of pure tungsten and molybdenum have already been given, while those for intermediate points are shown on the curves, Fig. 35.

The investigation of this series involved the treatment of more than 200 small ingots, and to give the details regarding each would be merely to repeat those already given for pure tungsten, which may be taken as

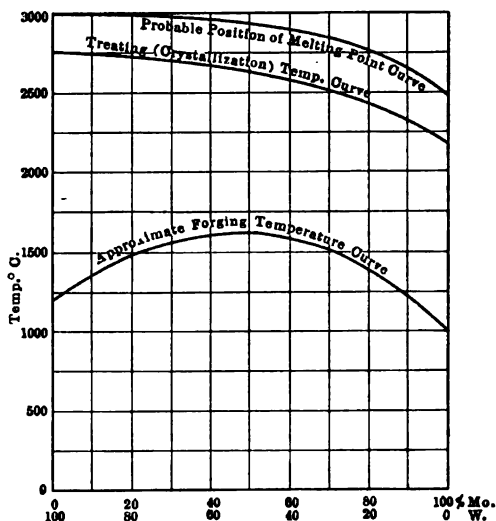


FIG. 35.—TREATING AND FORGING TEMPERATURES OF TUNGSTEN-MOLYBDENUM SERIES.

typical for any of these series. The formation of crystallites and crystal growth proceeds in a manner shown in the photomicrographs of the pure tungsten briquets, although—and here is an important point—the temperature representing a certain grain size changes with the relative amounts of the components, as the curve clearly shows. This is to be expected, and must be observed, in the treatment of the ingots; but it is not known whether this factor has received consideration in any attempted commercial production of such alloys.

A factor of equal importance is that of a corresponding change in the temperature of forging. If, for instance, only 1 per cent. of molybdenum were added to the tungsten, a treatment identical with that for pure tungsten would produce grains of such large size that even after repeated

forging (and this requiring a higher relative temperature) the material would still be unreliable. On the other hand, the addition of small amounts of tungsten to molybdenum would necessitate higher treating and forging temperatures, as shown in Fig. 35.

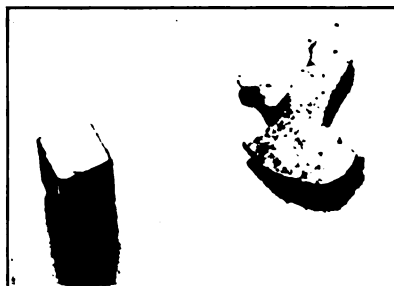


FIG. 36.—RAW AND FORGED BRIQUET OF 50 PER CENT. TUNGSTEN-MOLYBDENUM ALLOY.

Microscopical examination of these treated specimens revealed nothing that would indicate that these two metals formed other than a complete series of solid solutions; and if the crystallization curve may be taken as paralleling the melting-point curve (which assumption seems

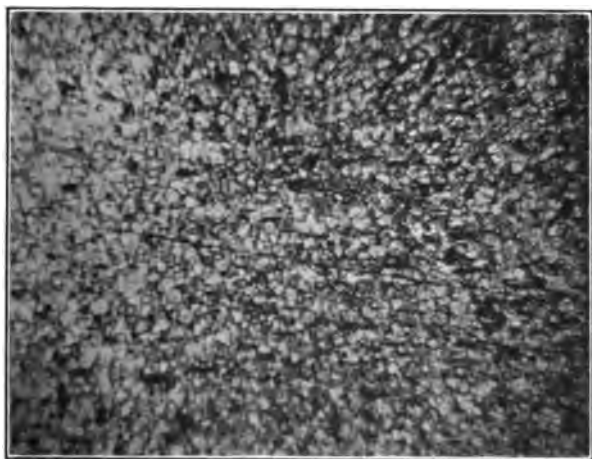


FIG. 37.—MICROSECTION OF BRIQUET, FIG. 36, AFTER TREATMENT AND BEFORE FORGING.

not illogical), then this latter curve will be found, when determined, to occupy a position approximately as indicated in Fig. 35.

Fig. 36 shows a specimen of the 50 per cent. alloy as the raw briquet and after being forged; while Fig. 37 shows a microsection of this same specimen, after treating and before forging.

VI. SUMMARY AND CONCLUSIONS

With regard to the degree of accuracy with which temperatures could be measured in these experiments; it must be pointed out that the object was not to establish these critical points for direct transference to any commercial plant (for different types of apparatus would necessitate a determination of these conditions to suit each individual case), but to determine their existence and influence. It would also be of no avail to locate these critical ranges because every different set of apparatus and conditions would require a new standardization.

In these experiments, however, the temperatures necessary to produce a certain degree of crystallization were considered as being located with a fair degree of accuracy, insofar as this may not be qualified by the existence of working conditions which were far from ideal.

As to the measurement of forging temperatures, no claim is made for more than close approximations, for this was properly not a one-man operation, and was performed by the writer with one eye to the optical pyrometer the other on the milliammeter scale; one hand on the pyrometer rheostats, the other using a hammer on the upper electrode; while the heating current was controlled by one foot on a lever regulating the transformer and rheostat.

The experimental work resolved itself into three parts, each being marked by a different method of attack, necessitated by limitations encountered as the work progressed under previously adopted methods.

The first part consisted of experiments on binary combinations of those of the metals which it was feasible to consider, and the melting points of which lay within the limits of ordinary fusion methods. The results of these experiments, performed as indicated therein, lead to the conclusion that metals or alloys of metals outside of the precious-metal groups, are unsuitable as substitutes for platinum.

The gold and silver alloys of palladium have been found to be excellent substitutes for platinum in its softer forms, and while not so chemically resistant, fill all requirements where conditions are not too rigid.

The second part develops the fact that except in two respects, pure ductile tungsten, and, to a lesser degree, molybdenum, meet all of the specifications of a practical substitute for platinum and its alloys. These two defects are its ease of oxidation, and the difficulty with which it can be soldered; and they have been overcome by coating with a precious metal or alloy, the resulting material being in many ways far superior to platinum or its alloys.

This material has met with instant demand, is in many cases replacing the best platinum-iridium alloys, and permits the performance of work which has been impossible with the materials hitherto available.

The third part describes the theoretical and practical considerations

involved in the manufacture of wrought tungsten and molybdenum, and gives results of the proper application of a similar method in the laboratory production of their alloys.

Wrought tungsten and molybdenum were produced on a laboratory scale, but no success attended the attempted production of alloys of tungsten with gold and palladium; while on the other hand, the alloys of the tungsten-molybdenum series were produced in wrought form. These operations were governed entirely by metallographic control, and their success suggests the possible application of a similar method in a treatment of such metals as iridium, tantalum, rhodium, osmium, etc., in combination with each other, or with tungsten or molybdenum, which may result in the production of alloys possessing properties far superior to those of any material now available.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Control of Petroleum and Natural Gas Wells

BY ALFRED G. HEGGEM,* M. E., TULSA, OKLA.

(New York Meeting, February, 1916)

It is the purpose of this article to describe methods recently introduced into the oil and natural gas industry to safeguard the lives of the workmen and to protect property from destruction. Only such drilling methods are described as pertain directly to the operations necessary to maintain the well under complete control at all times.

Wells Formerly Drilled Without Controlling Equipment

It may seem incredible that practically all wells, until recently, were drilled without the provision of any means for control. To drill a hole into a boiler carrying high-pressure steam without first providing means to control the escape of steam would be unthinkable, yet in oil fields wells are drilled into a gas or oil formation with a possible pressure of 300 to 1,800 lb. per square inch and a daily flow of 20,000,000 to 80,000,000 cu. ft. of gas or 1,000 to 15,000 bbl. of oil per day. Some wells have produced in excess of 100,000 bbl. of oil per day, but these are exceptional cases. In addition to the loss of product, one must consider the great danger attendant upon the promiscuous liberation of such an amount of explosive and inflammable substance.

Introduction of Metal Casing for Deep Drilling

The introduction of metal casing to secure a permanent and impervious wall in a well made the drilling of deep wells possible and removed many of the original problems encountered in well drilling, but it did not insure the control of the well. To obtain the full value of the casing it was essential that a pressure-tight joint be made between the wall of the well and the lower end of the casing. Many materials in various forms were used, including cotton, hemp, stable refuse, oats, rice, cloth, leather, cork, rubber, lead, clay and cement, resulting, after a slow process of evolution, in the general adoption of two types of bottom fittings for

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casing. These are the "long shoe," having a diameter approximately that of the casing couplings, and fitting into a hole especially prepared to insure a solid and close contact; and the rubber "packer," consisting of a rubber cylinder forced into tight contact with the wall of the well and the exterior of the casing respectively.

The other materials mentioned are used now only under special conditions, except clay and cement, each of which is used extensively, generally in combination with either the long shoe or with a packer.

Under general conditions, the weight of the casing, together with the friction between the casing and the walls of the well, is sufficient to withstand the lifting force of the gas or oil. The friction between the casing and the walls of the well is likely to vary greatly, and is uncertain at best. Additional friction is often obtained by cementing the casing within the well, and where very high pressures are encountered it is customary to secure together the several strings of casing by means of clamps, thus adding the weight of the outer strings of casing, as well as the friction, to that of the inner string.

Another practice is to bury heavy wood sills some distance below the derrick floor and, by means of heavy anchor bolts and suitable clamps, make connection to the top of the casing.

Gate Valve Does not Insure Control

With the bottom of the casing securely in place, and forming a pressure-tight, non-leaking joint, the top of the casing remains to be considered in the matter of control. Valves of various types were tried, and while not accomplishing as much as was desired, they were in many cases a necessity.

In its highest form, this type of equipment consisted of a gate valve, of inside diameter greater than that of the casing, surmounted by the casing head, the two being connected by a short nipple. This arrangement provided the usual casing head to which flow connections could be made and through which the usual drilling operations could be conducted. In addition, the gate valve furnished means by which the well could be shut in when the drilling tools were removed.

It was thought by many that a gate valve so placed on the head of a well insured control, but in practice many failures of this arrangement demonstrated that it did not safeguard either life or property and really caused a false feeling of safety. In fact, because of inability to close the valve promptly at a critical moment, oil or gas has become ignited and burned with great violence, making approach to the well impossible. The gate valve had to be removed by shooting off with a cannon ball, or other means, before the well, thus supposedly safeguarded, could be brought under control.

Requirements for Efficient Controlling Device

A proper closing and controlling device for the head of an oil or gas well must meet the following requirements: It must permit all drilling operations to be carried on without interference; permit of immediate and tight closing; control the flow without back pressure; insure safety to workmen; be simple in construction; compact in size; sufficiently strong to control maximum pressures; proof against injury in handling; unaffected by sand; and unaffected by fire.



FIG. 1.—OIL WELL FITTED WITH FLOW LINES BEFORE "DRILLING IN." GATE VALVES ARE PLACED ON EACH BRANCH OF THE FLOW LINE BUT THE WELL ITSELF IS WITHOUT MEANS OF CONTROL. THE COMMON CASING HEAD IS CAPPED WITH A SEPARATE TOP WHEN FLOW PRESSURE SUBSIDES SUFFICIENTLY TO PERMIT.

The Control Casing Head

The "control casing head," combining the functions of a gate valve and a casing head, was designed to meet these requirements, which are considered necessary to safeguard life and property during the operations of well drilling.

This device is similar in general appearance and size to the common type of casing head in general use (Fig. 2). It can be placed above or below the derrick floor, at the will of the operator, and is arranged to

receive the standard fittings commonly used with casing heads. The top opening is threaded to receive a drilling nipple or other top connections usually employed in gas wells.



FIG. 2.—THE CONTROL CASING HEAD.

The interior of the head is bored out to a true cylindrical form into which is closely fitted the plug or valve (Fig. 3). This valve is open at one end to provide a lateral passage for the oil or gas; the other end is



FIG. 3.—DETAIL OF VALVE PLUG AND BODY.

reduced in diameter to form a stem, which extends through a suitable stuffing box, and by which the valve may be operated. On the stem side of the valve a flat surface, or flange, fits closely against the base of the

stuffing box, making a tight joint, thereby to a large degree relieving the stuffing box of duty in preventing leakage. The extending stem is hexagonal in form to accommodate a wrench, but a transverse hole through it provides a more convenient means of operating by use of a bar of iron, such as a bolt or piece of 1-in. pipe.

To provide for the convenient operation of the valve at a distance, when the casing head is below the floor or is otherwise not readily accessible, the end of the stem is bored out and threaded to take an extension of standard 2-in. pipe.

The back of the valve is broad enough to close completely either top



FIG. 4.—PHANTOM VIEW SHOWING TOP OPENING CLOSED TO DEFLECT FLOW INTO TANK. THE DRILLING LINE IS SHOWN IN POSITION IT ASSUMES WHEN VALVE IS CLOSED WITHOUT WITHDRAWING THE DRILLING TOOLS.

or bottom opening in the body, and provide sufficient lap to prevent leaking.

On each side of the back of the valve is a groove, or notch, of sufficient size to encompass the drilling line, sand line, or torpedo line. By this provision the valve, when closed, while completely shutting in any flow, does not injure the line.

By means of the end opening, as well as by recessing the back of the valve, the pressures within the casing head are to a large degree counter-balanced, making the operation of the valve easy.

The device is simple, consisting of but four pieces, and having but one

moving piece. It cannot be damaged by any of the usual drilling operations or by rough handling.

Stops are provided within the body to prevent the valve from being turned too far in either direction. With the valve in normal position but a quarter turn of the stem is required to close the top or the bottom opening, turning the flow into the tanks or entirely shutting in a well.

The control casing head is used in much the same manner as gate valves and common casing heads, but it has many special uses that have developed largely from experience with the great number in operation in various oil fields.



FIG. 5.—PHANTOM VIEW SHOWING WELL SHUT IN ENTIRELY.

The Control of Wildcat Wells

When drilling a well in unknown territory it is not possible to foresee what conditions may be encountered, and many wells so drilled have "gone wild" with great destruction of property and in a number of cases with loss of life.

Some such wells have ruined fields of good prospective value by letting water in on the oil in such quantities as to render further operations in that district unprofitable.

For such cases it is desirable to place a control casing head on the first string of casing landed in the well and maintain it there through all subsequent drilling operations. This insures that the well can be kept under control, and if at any time oil or gas is encountered unexpectedly

the well can be instantly shut in until provision is made for taking care of the yield. It is not necessary to withdraw the tools as the valve will close tightly around the line, so no time need be lost in bringing the well under control. Also, with the flow stopped or controlled, there is no force acting to expel the tools, so this danger is avoided.

When a new string of casing is set inside of the one carrying the controlling device, the string may be set with the top below the head in order not to interfere with the free action of the valve; or the control head may be set on the inner string by using a swage nipple; or a smaller head of suitable size may be used.

If it is desired to shut in gas between the two strings of casing, a packing ring may be screwed into the larger casing head and a gas-tight joint made on the outside of the inner casing.

This process may be repeated with each string of casing inserted in the well. Of course where several heads are used on the same well, the larger ones should be placed below the derrick floor in order that the last casing head may not extend to such a height above the floor as to interfere with the ready wrenching of the tools or limit the length of temper screw that may be let out.

Use of Control Head in Established Oil Field

When drilling in an established field the control casing head is placed on the last string of casing to be seated in the well. Since it is about the same height as the common casing head, it does not interfere with the wrenching of the tools or other drilling operations.

If a pocket of gas is encountered, the well may be instantly shut in without removing the tools, all fires may be safely extinguished and the boiler moved back to a safe distance, when drilling may be resumed.

On reaching the oil sand, if the well starts to flow it may be shut in, or if flow-line connections have been made the entire flow may be diverted to the flow tank. When the flow ceases, drilling may be resumed and continued without loss of time until the well again starts to flow.

If the well flows continuously, an oil saver is placed on the line immediately above the control casing head, while same is closed. When ready, the valve is turned to allow the oil saver to enter the seat in the top of the casing head. This may be accomplished without loss of oil, wells producing as high as 10,000 and 12,000 bbl. of oil per day having been drilled in this manner without any oil showing on the derrick.

The prevention of waste of oil in drilling means not only a saving to the operator, but also the reduction of the fire hazard to a minimum. The driller and "tool dresser" do not get drenched in oil and thereby risk fatal injury by fire, which under the usual conditions prevailing before the introduction of the control casing head occurred with appalling frequency.

Moving the boiler away from the rig did not remove all hazard, for cases are known in which a tool dresser, in clothes soaked with oil from a spraying well, has approached the boiler to try the water cocks and had the flames from under the boiler set fire to his clothing. Such accidents are almost always fatal.

The Control Head Useful in Shooting an Oil Well

It often happens that, while a well still flows or sprays oil continuously, the yield has reduced to such an extent that shooting must be resorted to in order to save the well. There are, in fact, many reasons for shooting a well, even though the flow pressure is such as to make the placing of the shot a hazardous undertaking.

In the history of the petroleum industry, one finds that frequently a well would flow during the placing of a shot, causing the shell to be ejected from the well to explode in the derrick, with frightful loss of life and property.

The use of the control casing head on the inner string of casing makes the shooting of any well a safe operation. The shell containing the "shot" is lowered until the top is below the casing head, when the valve in the latter is turned to close the top opening, and the oil, which under former conditions would spray into the air and be lost, is now all caught in the flow tank. The slight opening in the rope notch of the valve to permit the torpedo line to slip through is not sufficient to permit an appreciable amount of waste.

The valve is maintained in this closed position until the shell is landed on bottom and the torpedo line withdrawn. It is then opened to permit the introduction of the squib, or other means employed to explode the shot, and is kept open until the well has cleaned itself, when it may be closed in time to catch the "second flow."

While usual practice demands that the well be open during shooting to permit the well to cleanse itself of the torpedo shells and of the sand and other material loosened by the shot, yet by the use of the control casing head all of this may be deflected into a tank without putting a back pressure on the well. Not only will all of the oil be saved, but a more complete record of the effect of the shot will be found in the accumulations arising from fragments of rock broken off and thus trapped.

The introduction of the control casing head has made it possible to shoot wells of any size regardless of the amount of flow, and it has in practice demonstrated its value by checking the flow in a well which threatened to expel the shot, consisting of 20 qt. of nitroglycerine contained in two shells joined together. In this case, of course, the valve was turned to close the bottom opening and entirely shut in the well. This was done instantly and without injury to the torpedo line.

Application of Control Head to Gas Wells

While the name "casing head" would seem to imply a use limited to oil wells, the control casing head is equally useful in drilling gas wells, and is used in the same manner as in drilling wells for oil.

In many places it is customary to drill gas wells during daylight only and to allow the well to remain open during the night, thus wasting a large volume of gas and reducing the rock pressure on the gas that remains in the sand after the well is completed. With the control casing head on the well the gas can be shut in during the time when active drilling operations are suspended. Not only is the waste of gas stopped and a higher rock pressure insured, but also the fire hazard is practically eliminated.

When the well is completed the control casing head is utilized in the same manner as the common T fitting. The usual top and side connections are made to the head, with the advantage that in the event of leaky gate valves, or damage to the lines, the gas may be instantly shut in and repairs made without waste of gas or danger of fire.

After a gas well is completed, it is the usual custom to limit the flow of gas into a pipe line by partially closing the valve at the head of the well. This, in the case of a gate valve, leaves a crescent-shaped opening of great length and slight width, with the result that all of the gas in passing through is forced to rub the sides of the opening, causing rapid wear of the gate and seats and destruction of the valve. The control casing head, in contrast, gives a small round opening gradually increasing to an oval-shaped opening, thus presenting a minimum rubbing surface in contact with the gas and reducing the wearing action.

The tight closing of the valve in the control casing head and the freedom from leakage are very important advantages in securing long life and freedom from accidents.

Effect of Sand on Control Casing Head

Owing to the absence of recesses and projections within the body of the control casing head, there is no place in which sand can lodge to interfere with the action of the valve. The sand carried by flowing oil during drilling operations, therefore, has not in any way disturbed the easy operation of the valve.

While sand cannot lodge and interfere with the action of the valve, there is a possibility of injury from the cutting action of sand. To meet this condition the valve portion is made with a curved surface to deflect the flow gradually from vertical, in the casing, to horizontal, in the flow line. This curved surface is backed by a great thickness of tough metal which will for a long time resist the scouring action of the sand.

By thus gradually changing the direction of flow, no flat surface is exposed to the direct impact of the sand and cutting action is materially reduced.

Some operators prefer to cushion the flow when the well is making much sand. This can readily be done by screwing a short joint of pipe into the top of the casing head and closing the top of the pipe by a metal plug. Sometimes a plug of wood is driven into the cushion pipe to protect the metal plug from being cut by the sand.

This cushion pipe may be assembled and secured in place, while the casing head is closed, after which the valve may be turned to the side position out of the path of the oil and sand.

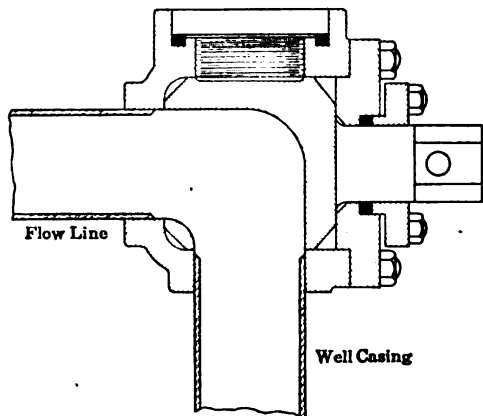


FIG. 6.—CONTROL CASING HEAD ON WELL SHOWING THE CURVED DEFLECTING SURFACE TO CHANGE GRADUALLY THE DIRECTION OF FLOW AND PREVENT THE MECHANICAL VAPORIZATION OF THE OIL. THE AREA OF THE PASSAGE THROUGH THE CASING HEAD BEING GREATER THAN THAT OF THE CASING INSURES A REDUCED VELOCITY AND PREVENTS FRICTIONAL BACK PRESSURE ON THE WELL.

The Control Head as an Aid in Killing a Gas Well

Gas is frequently encountered in drilling for oil and interferes with the further drilling of the well. Experience has shown that gas may be found in large volume and at high pressure above a formation containing oil in paying quantities. This fact has led to enormous waste of gas in an endeavor to reduce the pressure so that drilling could be continued to the deeper oil-bearing formation.

For a long time this waste of gas was considered necessary and was condoned on the ground that the value of the gas so wasted was less than that of the oil subsequently secured.

Aside from the loss due to the waste of the gas, the delay in drilling adds greatly to the expense of the well, and postpones the recovery of oil, entailing a further loss of oil through offset wells which have previously tapped the oil sand and are drawing from the common pool.

It behooves the operator to complete his well in the shortest time possible; therefore, the modern method is to seal off the gas sand and continue drilling without further delay.

The use of mud-laden fluid in killing a gas well is too well known to require a description here, but it may be interesting to know that the "lubricator" method of introducing mud into a gas well presented some practical difficulties in operation and was attended with some hazard.

The large gate valves which were used became difficult to operate, partly because they were not designed to operate in mud.

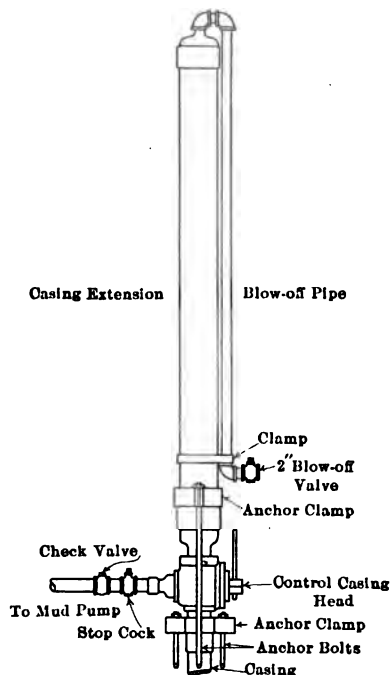


FIG. 7.—"LUBRICATOR" OF IMPROVED FORM FOR INTRODUCING MUD FLUID INTO A WELL TO "KILL GAS." ALL OPERATIONS ARE CONTROLLED FROM THE DERRICK FLOOR WITHOUT EXTRA HELP BEING REQUIRED.

By using the control casing head already on the well, these difficulties and the expense of buying and the loss of time in securing special gate valves are avoided.

On top of the control casing head are set one or more joints of any size casing available (two joints of 10-in. casing are my own preference). To the top of the casing extension thus provided, a 2-in. pipe line is connected and brought down the outside of the extension to within about 4 ft. of the floor, the end being fitted with a valve or stop cock (Fig. 7).

From the side outlet of the control casing head a 2- or 3-in. con-

nection is made to the pump discharge. This line should be provided with a check valve and a stop cock.

All fittings should be of suitable strength, and as the control casing head is designed for a safe load of 1,800 lb. per square inch all other fittings should be selected from extra heavy stock.

The 2-in. down pipe from the top of the extension should be securely clamped to the extension, and the valve-controlled outlet should be turned in a direction away from the operator.

These fittings are all made while the well is shut in. The pump is then started and mud-laden fluid pumped into the extension chamber until it is full, as indicated by mud showing at the outlet of the blow-off pipe. This outlet is then closed and the valve in the casing head opened permitting the mud to pass into the well. As soon as the extension chamber has emptied, the valve in the casing head is again closed and the outlet valve on the down pipe opened. This operation is repeated until the well is filled and the gas killed.

The advantage of this arrangement lies in the fact that everything is controlled from the derrick floor and no extra help is required. Also time is saved, since the pump will start delivering mud into the extension chamber as soon as the gas pressure is reduced, and, further, it is possible to pump directly into the well without any change of fittings.

Control Head on Flowing Well Reduces Oil Loss and Fire Hazard

With the old method of closing in a flowing well, a flat top was set in the old-style casing head several inches above the side outlet. The column of oil would strike this flat top at high velocity and be broken up into a fine spray, which, upon mixing with the gas, would appear as a blue "smoke" over the flow tank. This spray would float in the air for a considerable distance, wasting a large amount of oil and covering the surrounding trees and grasses with an inflammable oil. It would also settle in so-called "gas pockets" along the road, frequently becoming ignited by passing automobiles, with attendant loss of life and property.

As previously mentioned, the curved form of the valve in the control casing head gradually deflects the current of oil into the flow line. Therefore, the oil is not broken up into a spray, more good oil is put into the tank, and the fire hazard is reduced.

The Control Casing Head Now Largely Used

The control casing head fills a want that has been long felt and is rapidly being adopted as standard equipment in the drilling of wells for petroleum and natural gas.

At present (September, 1915) approximately 1,000 control casing

heads are in use, and, although a great variety of conditions has been encountered in drilling wells, ranging from dry holes to oil wells producing 12,000 bbl. a day, and gas wells having a rock pressure of 1,150 lb. per square inch and volumes in excess of 40,000,000 cu. ft. per day, there has not been a single failure of the head to safeguard life and property, and not one well has burned or become unmanageable.

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Economies in a Small Coal Mine

BY HERBERT A. EVEREST,* OKLAHOMA CITY, OKLA.

(New York Meeting, February, 1916)

THE idea of economical production is usually associated with large operations, tonnages, and mines, with even larger capital behind them. Nevertheless many small mines operate in the shadow of large competitors and make a good showing on the capital invested despite larger overhead expenses.

For the purpose of this discussion I will divide the cost of production into classes, and specify opposite each the approximate percentage expended thereon:

1. Labor, including miners and company men; 60 to 75 per cent.
2. Development, including all necessary yardage, room turning, crosscuts, etc.; 7 to 12 per cent.
3. Deadwork, covering payment to miners for handling falls, draw-slate, or faults, rock work, water, and in general, all nonproductive labor; 3 to 7 per cent.
4. Supplies: mine timbers, oil, brattice material, lumber, cement, and repairs to equipment; 2 to 6 per cent.
5. Expense: management, selling, office, taxes, etc.; 1 to 5 per cent.
6. Depreciation of coal reserves and royalties; 6 to 10 per cent.
7. Depreciation of equipment and interest on capital invested; 1 to 3 per cent.
8. Fuel; $\frac{1}{2}$ to $1\frac{1}{2}$ per cent.

The labor cost for a small mine is relatively much lower than for a large mine, particularly the sum paid to company men. Idle-pay expense is cut to a minimum; the labor item being larger than all the others together, a saving in labor makes a decided showing on the total cost.

The development charges in small and large mines are about the same. There should be a small showing on the deadwork item in favor of the small mine. The unit cost for supplies is usually small in the small mine. The operator of the small mine finds, owing to a limited tonnage, that the expenses of management, selling, etc., are abnormally high.

The item, depreciation of coal reserves and royalties, is usually about

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equal for both small and large mines; while the small mine is favored with regard to depreciation of equipment and interest on capital invested. If the mine is a shallow one, the fuel account is also in favor of the small mine—as a rule, not because of any great effort on the operator's part.

In the small mine the principal savings are in labor, supplies, equipment, and capital, because the working places are closer together, thus concentrating the working force. Fewer mine cars, less mine trackage, fewer entries and air courses are required and need be maintained, and the surface equipment is cheaper.

Concentration of the working force brings the mine labor under close supervision; and where fewer men are employed, all work to better advantage. The movement of cars is rapid and the miners load a good turn, producing a greater tonnage per unit of area opened. Concentration and close supervision result in the operation of the mine on a very small capital investment per ton of production, the charge for interest and depreciation of equipment being proportionately low.

Under close supervision supplies are not wasted, misused or lost, in many instances materials being used over and over again. Entries are usually worked out during the life of one set of timbers so that the expense of retimbering is obviated.

The operator of a small mine has a keen regard for the inroad on his profits that even one extra man on the payroll would produce. He realizes that much valuable time is wasted in the mine—some of it due to laziness, but more to misdirected energies and lack of or improper planning, so that the men are obliged to wait for material, tools, or cars. The chances are, in a small mine, that if a place is to be timbered, a fall is to be cleared up, or a room switch to be put down, all of the material is on the spot before the man arrives to do the work. The different working places are close together, therefore the workman does not waste much time in going from one place to another. In this way one tracklayer can do the work of two men who move long distances between jobs, and must look over the place, order their own material, and probably wait for its arrival.

At the rate of \$3 per day, and working 200 days in a year, a man receives \$600 per year; in 10 years he has received \$6,000. During that time he has been a liability, because of the possibility, through carelessness on his or some other miner's part, of an accident or fatal injury. Any mechanical device, which might supplant a man, the first cost and upkeep of which would be less than \$6,000 in 10 years, would be a good investment. The mechanical device will not lay off or quit; this precludes the necessity for training new men on the job, often a costly operation. The machine does not come under any workman's compensation act; it does not strike nor get out of repair as often as the human machine (provided it is properly designed and cared for).

Some of the comparisons made are unfair, but inasmuch as the small mine usually has a limited acreage and frequently shallow coal, it follows that the mine will never have many long entries, that the haul will be short and the work, while it lasts, will be concentrated.

Reverting to basic principles, the economies in a small coal mine result almost solely from close concentration of the working places and a still closer supervision of the labor employed.

Metallurgical Practice in the Witwatersrand District, South Africa

Discussion of the paper of F. L. BOSQUI, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 101, May, 1915, pp. 997 to 1033.

H. A. WHITE, Springs, Transvaal (communication to the Secretary*). —It is, as the author points out in his valuable paper, a fact, that in many plants the "trommel washings" are allowed to become eyesores, but the simpler system of pumping the entire product direct to the tube-mill circuit will avoid the extra expense of separation and small-scale treatment when one considers the relative distance of the points and the proportionately small amount of water.

It is a matter of record that in stamp batteries the increase of weight has been developed concurrently with the increased use of tube mills, but independently and on its own merits. The satisfactory duty, excellent running time, and freedom from breakages in the newer mills, such as the Modder Deep, with 2,000-lb. stamps, will probably go far in removing some of the conservative prejudices mentioned by Mr. Bosqui.

The success of the City Deep trial of the Nissen stamp cannot be entirely attributed to better feed distribution and discharge effects (published experiments on two-face discharge mortar boxes showed no advantages). Attention is directed to the treatment costs given by the author both before and after the increased capacity resulting from the addition of 16 Nissen stamps at Modder B. It will be observed that the only item of cost resisting the tendency to reduction caused by increased tonnage is that of milling, which has risen 1.3c.

With reference to the provision for amalgamation in the new plans, it seems obvious that a great loss of head would be eliminated by passing the tube cone overflow over a separate top series of plates; three at the top and three at the bottom will effect the maximum amalgamation. This plan is followed at the Princess Estate and Gold Mining Co., Ltd., Modderfontein Deep Levels, Ltd., and Geduld Proprietary Mines, Ltd., but elsewhere reliance is placed upon ample cone capacity and the underflow only is passed over three plates after going through the tube mill, while the overflow is passed direct to the cyanide works; both methods avoid the largely unnecessary elevation of considerable quantities of water, fine sand and slime involved in Mr. Bosqui's scheme.

The most economical use of the tube mill is still receiving considerable attention though many points have been eliminated from the field of controversy. The running speed favored by Mr. Bosqui (28 r.p.m. for the 5 ft. 6-in. tube) is very close to the point of maximum production of — 90 grade per horsepower-hour consumed. There is, however, another

* Received Nov. 17, 1915.

interesting maximum, which is that of capacity or greatest production of -90 material per tube. This speed, 31 to 32 r.p.m., is the more likely to be adopted as it involves but little waste of power, which is, after all, not the only item of cost.

The latest Osborne bar liner coming largely into use has the upright bar held in position on top of instead of between the flat bars, as illustrated in this paper. This allows longer life for the more expensive radial steel bars and the reduction in crevice capacity is reflected in the saving of amalgam caught therein. We are still looking for the ideal liner which must be cheap and durable, have a good grip on the pebble load, be easily and quickly renewed, and cause no locking up of gold in the form of amalgam, in crevices.

The tube-mill scoop has now passed the experimental stage and is giving satisfaction, but it would be interesting to know whence the author derives the "sloping line of discharge" from inlet to scoop. The level of pulp is surely as horizontal as in the old tubes, though the height is considerably reduced. The "whole hog" peripheral discharge had only one defender at the early date referred to.¹

The scheme presented for collection and treatment of sand in fulfilment of the author's undertaking to "evolve a simpler method than the filter table" suffers from even graver disadvantages than the other well-known efforts to eliminate the table itself (the patented feature) from the whole system used under that title.

Mr. Bosqui's idea of treating a charge with cyanide solution in the same vat in which it is collected by water, re-introduces the identical defect which finally caused the abandonment of double treatment on the Rand, referred to by Mr. Bosqui. In that case also, the consequent contamination of the mill water by cyanide, and the inevitable, continuous loss of gold was aggravated by the occasional accidental heavy losses caused by allowing the cyanide solution to run into a vat still being filled by water-borne sand. These losses were doubtless better known than advertised; nevertheless the re-introduction of such possibilities is a retrograde step in defiance of the more modern practice which encourages only one method of getting cyanide solution into the mill water-stream, *i.e.*, with a bucket. The new plants of the Barnato group now working, such as the Van Ryn Deep, Ltd., keep the collecting and treatment vats separate and double wash the sand to free it from colloidal slime, getting a better effect from the added water than can be possible in the single-stage hydraulic separation advocated in this paper.

Why this conservatism with regard to the sand filter table? True,

¹ *Journal of Chemical, Metallurgical and Mining Society of South Africa*, vol. vii, p. 371 (1906-1907).

it is a local invention, yet this method is coming into extended use, and experience on mines such as the East Rand, the Princess Estate, the Geduld Proprietary, the Modder Deep, and the City & Suburban Mining & Estate Co., Ltd., warrants a belief in its future. At the Geduld and Princess the conversion of old plants enabled a true comparison to be made and the improvement in sand residues in strictly comparative experiments on a working scale of 0.05 dwt. at the Princess was much more than confirmed at the Geduld, where there was a greater margin to work on.

With the sand filter table system the dewatered sand is pumped in a stream of weak cyanide solution to the collecting vat where the treatment is also finished and every particle of sand must come into contact with cyanide solution, which is not inevitably the case with any other method. Thus we get the advantages of a preliminary "weak wash" which formerly had to be discontinued because of the time lost, while now the sand is under treatment half an hour after it has left the tube-mill plates.

As far as capital cost is concerned, even with treatment in collecting vats, no great saving can possibly be shown if it be recollected that a longer time of treatment will of necessity be required with a water-collected charge, so that more vats must be installed than are required in the safer filter table method. As regards cost of maintenance and operation, a reference may be made to an extract from the official report on the filter table at the City Suburban.² The excellent results there shown with regard to light cost and increased extraction serve to confirm the general experience and justify the adoption of the sand filter table system on the fine new plant at the Modderfontein Deep, the most recent on these fields. In this last instance, of course, no comparison with older systems is possible, but the results obtained were better than anticipated from the preliminary small-scale tests made.

The Dorr collecting vat for slime may have the attributed disadvantage of not being a local engineering product, but so far the real reasons for its absence on modern Rand plants are as follows: (1) The increased capacity for a given area is small, as pointed out by Mr. Bosqui. (2) The extra moisture contained in the underflow involves a loss of about 2c. per ton for cyanide alone (required to bring this extra water up to treatment strength). The figure given by the author for moisture in settled slime in the ordinary collector (50 per cent.) is much too high, and modern plants easily secure a reduction to 40 per cent., and still lower figures are obtained when much very fine sand is present. The use of 70-ft. vats in place of the 56-ft. mentioned is also becoming general in pursuit of the successful "large unit" policy. (3) The increased capital

² *South African Mining Journal*, vol. ix, Part II, p: 739 (Jan. 27, 1912).

and running costs of the Dorr vat have insufficient advantages over the simple method now in use.

The Butters filter system, for the introduction of which our thanks are largely due to Mr. Bosqui, is highly appreciated and would have much more extended use if the slime here were generally of higher value than it is. The extra cost is balanced only by the higher extraction when the original slime value is in the neighborhood of 2 dwt. Take for example the Princess Estate where the average value of the residue is under 10c. and many other mines where it is under 20c., the undissolved gold accounting for more than half of this. In such cases it is clear that the quoted running cost of 6c. to 8c. will leave little margin to pay for the extra capital cost involved. This process is very successful at the Geduld with slime of initial value of 3.5 dwt. and where, by the way, the only continuous use of air-lift vats on *current* slime is as yet practiced, beside the new plant at the Modder Deep. The East Rand Proprietary Mine vats are used for accumulated slime at present.

The economy in zinc consumption claimed for the Merrill press is not warranted by the figures quoted (0.152 and 0.154 lb. per ton of solution). The average figure for the Rand is about 0.35 lb. per ton of pulp and on some plants 0.25 lb. is maintained. This worked out on a $2\frac{1}{2}$ to 1 pulp, the usual ratio, would be 0.14 lb. and 0.10 lb., respectively, per ton of solution. Any economy per ton of pulp shown by the Merrill system must therefore be offset by reduced treatment, and the figures given for gold value of the residual solutions, being distinctly poorer than those obtained with zinc shaving, contribute to the impression that this process, though not without valuable features, will have to seek a welcome elsewhere.

While it must not be overlooked that the presence of zinc increases the stability of cyanide solutions to an extraordinary degree, there is the possibility of adopting some new method if it can show greater security of gold, ease of clean up, and reduction of space, combined with practically complete extraction of the gold value from very dilute solutions.

It has frequently been a matter of regret to the Rand metallurgists that processes and devices apparently successful elsewhere, have proved unsuited to local conditions. Although fresh information of local or oversea origin and development will always be welcome, the present, "conservative" attitude of preferring actual trial to glowing verbal accounts will continue manifest in Rand metallurgical practice.

Gasoline from "Synthetic" Crude Oil.

Discussion of the paper of WALTER O. SNELLING, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 100, April, 1915, pp. 695 to 704.

A. C. McLAUGHLIN, San Francisco, Cal.—It seems to me that Mr. Snelling's work, if it is what it appears to be, is one of the most important contributions to petroleum technology that has been brought out in the last 25 years. If his end products are what they seem to be, he has made a discovery, which, in my judgment, will revolutionize the petroleum industry.

As to industrially carrying out these processes I do not think there is the slightest difficulty, for I have had some work recently where crude oils were handled successfully at a temperature of 750°F. and pressure of 750 lb. per square inch.

WILLIAM A. WILLIAMS, San Francisco, Cal.—What metal did you use, Mr. McLaughlin?

A. C. McLAUGHLIN.—Ordinary seamless tubing, extra strong.

WILLIAM A. WILLIAMS.—Tests were made in Dr. Rittman's experimental plant at Pittsburgh, upon samples of residuum furnished by Mr. Bell, the residuum being the remainder of the crude oil after the lighter fractions had been removed.

After the material had been run through Dr. Rittman's plant, yields of from 43 to 58 per cent. of material, boiling under 150°C. were obtained. The distillate had a peculiar odor, due to the presence of benzene, toluene, and other unsaturated hydrocarbons.

W. N. BEST, New York, N. Y.—What is the difference between the Rittman and the Snelling process or system?

WILLIAM A. WILLIAMS.—The fundamental difference is that reactions take place under relatively higher temperatures and pressures and with none of the oil present in its original or liquid state, while the reactions secured by Mr. Snelling and other investigators have been from the liquids in the presence of some vapors.

Under such conditions as Mr. Snelling operates, the temperatures and pressures used are limited and the flexibility in the control of these variables is extremely limited when compared with the Rittman process.

In making gasoline, Dr. Rittman has found favorable temperatures around 500°C. and pressures around 150 lb. to the square inch. Dr. Rittman has recovered from a 300° distillate as high as 85 per cent. of the original product which contains more than 30 per cent. which will boil under 150°C.

A. C. McLAUGHLIN.—What strikes me as most important, in connection with this Snelling discovery, is that it is so different from all cracking processes worked on during the past 30 years. The process of cracking is not new, and my experience as a refiner taught me that the cracking of oil was analogous to what is accomplished with the microscope, *i.e.*, there is a gain in magnification but a loss in definition; what is gained in the stills is lost in the agitators. Compounds are formed which must be taken out with sulphuric acid.

Now, it appears that Mr. Snelling has produced low-boiling, saturated compounds. If he has done that, he has done something never achieved before, and I regard it as a big advance.

M. L. REQUA, San Francisco, Cal.—Has anybody made any estimate of the cost?

A. F. L. BELL, San Francisco, Cal.—In connection with the tests of which Mr. Williams speaks, I do not know that any estimates have been made of what a practical working plant would cost, since the process is in the experimental stage.

The samples Mr. Williams returned to us were tested at our refinery laboratory and re-run as we do our ordinary tops. The highest distillates that came off, or that we could get off, were about 50°Bé., but they had the same boiling points and corresponded closely, by the Engler test, to the 60°Bé. gasoline refined from crude oils and marketed here. But when the samples were treated with sulphuric acid in the same manner as our ordinary distillates, there was a marked reaction; the sulphuric acid turned the distillates black and threw down an excessive sludge, making it impossible to clean them with sulphuric acid. Clarifying with lye and by filtration was only partially successful. We obtained products of a beautiful red color, and some with green hues. But our samples were so small (only 4 oz.) that we could not determine what process would make them a commercial product. I do not think, however, that they can be treated in the same manner as the ordinary distillates of petroleum. They do not have any of the repugnant odors that the ordinary distillates of petroleum have, when heated to the same temperatures in the ordinary still under atmospheric pressure.

WALTER STADLER, San Francisco, Cal.—In reading over the literature which has come to my notice regarding the Snelling system, it has been observed that most of the oils that have been subjected to the treatment and on which detailed results have been published are those from the eastern United States and from Oklahoma. The crude oils from these regions, as I understand them, are usually composed of saturated hydrocarbons of the paraffine series, or a mixture of such saturated hydrocarbons with unsaturated hydrocarbons of the olefine series.

According to the results noted by Mr. Snelling, the process must virtually consist of converting higher-boiling heavy hydrocarbons of both the paraffine and olefine series into lighter low-boiling hydrocarbons of the paraffine series, since by repeated treatment a crude oil resembling the original oil is each time produced from which low-boiling members without color or odor can be distilled.

In California we are naturally interested in the California oils. As yet, detailed results from California oils subjected to the Snelling process have not been noted by me in the literature put forward. In this regard I may have overlooked something. The bulk of our California crude is entirely different than the Eastern and Oklahoma oils in that it is composed of unsaturated hydrocarbons to a degree of unsaturation frequently greater than in the olefine series, of naphthenes, some of the aromatic hydrocarbons, and minor amounts of the paraffine hydrocarbons.

If from the complicated California oils repeated treatment of the heavy residues will each time yield results comparable with those obtained from the Oklahoma and Eastern oils and give lighter products without color or odor, it will mean considerable for the technology of petroleum. It will mean, at least in part, a virtual reversal of the old cracking process. It will mean a process in which the more stable lower-boiling bodies are produced without the use of catalyzers. Results with California oils will therefore be of considerable interest.

A. F. L. BELL.—Our California oils are of asphaltic base, whereas most of the Eastern oils are of paraffine base, but I am under the impression that California oils can be treated as well in the Snelling process as the Eastern oils. I simply judge that from the treatment of the California oils in the Rittman process. Mr. Williams informed me that he was able to obtain a higher percentage of gasoline from the California oils than from the Eastern oils.

WALTER STADLER.—Were not those samples treated with sulphuric acid, and did you not get large quantities of tar which would indicate cracking?

A. F. L. BELL.—Yes.

DAVID T. DAY.—I would like to mention two points: First, concerning the specific gravity of the distillates, it seems odd to an Eastern man to obtain gasoline, by means of the Rittman process, the specific gravity of which is very high. The explanation is simple: When these oils crack and as the temperature goes up, more and more of the aromatic series are produced. If the temperature is kept low, so that the oil is just cracked, a different specific gravity is obtained. The specific gravity of all these cracked products is high as compared with neutral gasoline.

Second, the matter of the purification of these cracked distillates is one of great importance. Such a large proportion of unsaturated bodies is present in the cracked gasolines that it would be out of the question to purify them by the ordinary methods employing sulphuric acid or alkali. Within the last 30 days I have been developing a process for refining the Rittman gasolines which is absolutely satisfactory, simple and cheap. So far I have not used any sulphuric acid. The refined product may not be as sweet as ordinary gasoline, but it is plenty sweet enough.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Effect of Aeration and "Watering Out" on the Sulphur Content of Coke

BY J. R. CAMPBELL,* P. S., M. S., SCOTTDALÉ, PA.

(New York Meeting, February, 1916)

In order to discuss the subject intelligently, it will be necessary to touch briefly on the forms in which sulphur is supposed to exist in coking coal to be carbonized in beehive or byproduct ovens.

Sulphur is known to exist in coal as sulphides and sulphates, as can be determined experimentally. Then there is the so-called organic sulphur, i.e., sulphur in combination with the carbon, hydrogen, and oxygen of the coal, about which much has been written, but nothing definitely proven in an experimental way. In fact, so far as we know, there never has been developed a satisfactory and conclusive method in the laboratory for the direct determination of organic sulphur.

For most coking coals, it can be safely assumed that the preponderance of sulphur is in the form of pyrite (FeS_2). When exposed to comparatively low temperatures during the coking process, it loses its sulphur according to the following chemical reaction: $7\text{FeS}_2 = \text{Fe}_7\text{S}_8 + 6\text{S}$. It will be seen, therefore, that six out of the 14 atoms of sulphur (or 42.8 per cent.) are expelled, or volatilized. Fe_7S_8 remains in the coke as pyrrhotite, or magnetic sulphide of iron; this accounts for the highly magnetic properties of the powdered coke. A more commonly accepted reaction is $\text{FeS}_2 = \text{FeS} + \text{S}$, in which the straight sulphide of iron is produced with 50 per cent. volatilization of sulphur.

In the beehive methods of coke making, air is introduced in sufficient amounts to carry on the distilling and the coking processes, and the sulphur is oxidized along with the other volatile products: First, into sulphur dioxide (SO_2), known by the pungent and suffocating odor emitted from the trunnel head; second, into sulphuric anhydride (SO_3); and, finally, into sulphuric acid, as it comes into contact with the air and moisture. In a properly regulated draft on a beehive oven, there is never complete combustion of the gases. In other words, the coking process should be carried on in a reducing atmosphere, and a low-grade producer gas kept issuing from the trunnel head. For coals of about 30 per cent. volatile matter, the ratio of air to gas is $3\frac{1}{2}$ to 1. In complete combustion, the ratio is 6 to 1, producing an extremely high temperature

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in the crown of the oven (3,500°F.) more than enough to cause fusion of the refractories used. A good coking temperature is 2,500°F. in the crown of the oven.

It follows then that the sulphur remaining in the coking mass as iron sulphide cannot in any way be affected by the aeration or draft on the oven. Among beehive coke-oven operators there used to be a saying, "the hotter the oven, the more sulphur burned out." In view of the foregoing, this probably never was true unless the aeration was carried to complete combustion of not only the volatile gases but the fixed carbon itself, in which case the iron sulphide (FeS) would, of course, be oxidized to Fe_2O_3 , and the sulphur liberated, as usual. This condition would not be productive of good results, as the percentage of coke yield would be abnormally low and the "ashes," or "braize," correspondingly high. This is evidence that so far as the pyritic sulphur is concerned, any sulphur that is volatile at all is so at comparatively low temperatures through the agencies of distillation, and no practical method of supraeration will help in the least in the further elimination of sulphur.

On the other hand, overheating promotes certain chemical reactions in which sulphur forms compounds with other bodies on which heat has no effect. If the coking process proceeds too quickly, or if the heat is irregular, the desulphurization of the coke is apt to be less complete. In these matters, much depends upon the skill and judgment of the burner.

In the byproduct oven, which is essentially a true distillation process, the heat being applied externally, and no air supplied to the oven itself, the sulphur is distilled from the pyrite as atomic sulphur, as previously explained, but, apparently, it afterward combines with the hydrogen of the coal gas, forming sulphuretted hydrogen (H_2S), for it is necessary to remove this very objectionable gas before artificial gas can be used for domestic purposes. This is accomplished by the use of hydrated oxide of iron, a byproduct now obtainable from mine-water purification.

Here again appears the fallacy of supraerating beehive ovens to eliminate sulphur, for in byproduct ovens in which no air is admitted there is just as much desulphurization of the coke as in the beehive process; in fact, there is more desulphurization when the matter of increased yield is considered. For example: Given a yield of 70 per cent. coke by the beehive process and 75 per cent. by the byproduct process from coal carrying 1 per cent. sulphur, the respective problems are as follows:

	Beehive Oven, Per Cent.	Byproduct Oven, Per Cent.
Sulphur in coal.....	1.000	1.000
Sulphur volatilized.....	0.428	0.428
Sulphur remaining.....	0.70)0.572	0.75)0.572
Sulphur in coke.....	0.817	0.763

Thus the practical volatilization of sulphur from the beehive process is only 18.3 per cent., while in the byproduct it is 23.7 per cent. In our own beehive practice we have always used 20 per cent. as the amount of sulphur eliminated from all sources, figuring from coal to coke, this being the result of numerous laboratory and practical tests on the Connelsville coal.

Attention has been directed in the preceding paragraphs to the sulphur as pyrite. Let us look into the other forms that may be present in addition to the sulphide. Any organic sulphur present remains, for the most part, in the coke. It is readily seen how it would be unaffected by aeration except in event of total destruction of the carbon itself, with which the organic sulphur is supposed to be combined.

Any sulphates present, such as CaSO_4 , would also be unaffected by aeration or supraeration in the beehive process; no amount of "airing" would help to eliminate the sulphur. We know of cases in which the sulphur in the resultant coke was as high, or even higher, as in the coal, from which it must be concluded that much of the sulphur was present in the coal as sulphate, or the so-called organic sulphur; or else the ash of the coal was rich in iron, lime and magnesia, for there is a dictum that it is never possible to produce a low-sulphur coke from the coal the ash of which is high in these elements. Supraeration cannot and will not ameliorate these conditions.

The Effect of Quenching on Sulphur

Perhaps everyone is familiar with both the beehive and byproduct oven practice in this country, quenching the coke, by copious amounts of water. In the former case, it is customary to "water out" in the oven itself, while in the latter it is done externally by means of suitable quenching stations. In beehive practice, it takes approximately 1,000 gal. of water to quench a 5-ton charge of coke, consuming about $\frac{3}{4}$ hr., either by hand watering with a hose, or with a sprinkler of the Stauff type, which is placed in the oven on top of the coke. The byproduct practice is much quicker, but the external watering produces darker coke.

Theoretically, it would seem possible to remove considerable sulphur by the quenching process. We have already indicated how iron sulphide (FeS) is formed in the coking process. The action of the water on the sulphide is as follows: $\text{FeS} + \text{H}_2\text{O} = \text{FeO} + \text{H}_2\text{S}$, in which it is assumed that the coke is quenched externally. The odor of sulphuretted hydrogen is easily detected (but it will be appreciated that a little of this gas goes a long way), and the color of the coke, wherein rust spots appear, shows the presence of iron oxide. The practical coke burner is always suspicious of what he terms "rusty coke," as it invariably is an indication of high-sulphur coke. The same holds true

in quenching in the oven itself, as in beehive practice. Sulphuretted hydrogen (H_2S) is evolved, but, at the same time, the water as steam is decomposed by the carbon, $H_2O + C = CO + H_2$, which fact probably accounts for a part of the large percentage of carbon monoxide and hydrogen gas found in the gas from trunnels of beehive ovens, the water being formed from the combustion of the gases during the coking processes, which is virtually a low-grade producer gas of the following composition:

	By Volume, Per Cent.
CO_2	3.0
CO	9.0
H_2	11.0
CH_4	0.3
N_2	76.7

The desulphurization of coke by water cannot be complete. The coke mass cools too quickly and its very structure prevents the rapid penetration of the water thrown on it, especially in the denser varieties. As the temperature is lowered the reaction involved becomes too slow to be of practical benefit. Data as to the exact amount of sulphur eliminated in this way are rather scarce, but our own experience, based on the general laws of volatilization set forth elsewhere, leads us to the conclusion that only an infinitesimal percentage of sulphur is thrown off during the quenching process. Laboratory and practical tests, in which the coke has been allowed to cool naturally, show but little difference in the sulphur content from those in which the coke has been quenched with water. Certainly there could be only a few hundredths of 1 per cent. in favor of the water-quenched coke, at the most, in a coke averaging 1 per cent. sulphur.

The prime object of the use of water is to lower the temperature of the coke for handling, not for desulphurization, and the quenching process should be so regarded. However, we are of the opinion that quenching by the byproduct practice will eliminate more sulphur than by the beehive method, which may partially explain the greater total volatilization of sulphur in the former, elsewhere intimated.

It may not be generally known, in connection with the subject, that addition of muriatic acid (HCl) to the water greatly facilitates the removal of sulphur during the quenching process. The action of this acid on iron sulphide is positive at all temperatures, thus, $FeS + 2HCl = FeCl_2 + H_2S$. There was, of course, a time when the cost prohibited the use of this method, but with the depletion of our low-sulphur coking coals this may, in the near future, be a factor in the elimination of sulphur.



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 Steptoe Valley Smelt. & Refin. Co., McGill, Nev.
 Nevada Consolidated Copper Co., McGill, Nev.
 Ray Consolidated Copper Co., Ray, Ariz.
 Chino Copper Co., Santa Rita, New Mexico
 American Smelt. & Refi. Co., Monterey, N. L. Mex.
 Esperanza Mining Co., El Oro, Mex.
 Braden Copper Company, Braden, Chile, S. A.
 Federal Lead Company, Flat River, Mo.
 Old Dominion Copper Mining Co., Globe, Ariz.
 Copper Queen Consoli. Mining Co., Bisbee, Ariz.
 Detroit Copper Co., Morenci, Ariz.
 Stag Canon Fuel Co., Dawson, New Mexico.
 Belmont Milling Co., Millers, Nev.
 Desert Power & Mill Co., Millers, Nev.
 Tonopah Belmont Develop. Co., Tonopah, Nev.
 El Oro Mining & Railway Co., El Oro, Mex.
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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 110

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1916

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

VOLUME LI IS ISSUED

Members are advised that Volume LI of the Institute's *Transactions*, which contains the proceedings of the New York Meeting, February, 1915, is ready for delivery. Copies have been sent to each member whose dues for the current year are paid, in accordance with the usual custom. If any member does not receive it in due time after the payment of his dues will he kindly promptly notify the Secretary of the Institute.

Come to the New York Meeting. See announcement on following page.

NEW YORK MEETING

Full details of the program for the New York meeting were given in the January Bulletin and the announcement mailed to members.

The following brief synopsis is offered simply as a reminder of the principal events of this meeting which Chairman Browne says is going to be "different."

Time.—Monday, Feb. 14, to Thursday, Feb. 17, 1916, inclusive.

Place.—Engineering Societies' Building, 29 West 39th St., New York.

Events.—Monday, Tuesday and Wednesday (Morning and Afternoon):

Technical Sessions. Tuesday, 10 a. m.: *Annual Business Meeting.*

Luncheon served in the Engineering Societies Building between sessions each day.

Monday evening: *College Night.*

Tuesday evening: *Smoker and Entertainment.*

Wednesday afternoon: Visit to Senator Clark's art galleries, followed by tea at the residence of Mrs. Bradley Stoughton.

Wednesday evening: *Annual Dinner*, followed by dancing.

Thursday. *All-day Excursion* aboard a Navy boat with visits to Brooklyn Navy Yard and Sandy Hook Proving Grounds, by courtesy of the Navy and Army Departments, United States Government.

For the Ladies

Luncheon each noon at the Engineering Societies Building;

Monday afternoon: *Thé Dansant.*

Tuesday afternoon: *Hippodrome*, to see fancy ice-skating.

Wednesday afternoon: Visit to Senator Clark's art galleries, followed by tea at the residence of Mrs. Bradley Stoughton.

Wednesday evening: *Annual Dinner and Dance.*

Thursday. *All-day Boat excursion.*

REPORT OF THE SECRETARY OF THE AMERICAN INSTITUTE OF MINING ENGINEERS FOR THE YEAR 1915

I have the honor to submit herewith the report of the Secretary for the year 1915 showing the principal activities of the Institute:

Meetings.—The 110th Meeting, including the Annual Business Meeting, was held in New York City, Feb. 15 to 17, 1915. The full report of this meeting is given in Volume LI of the *Transactions*. The number of papers presented was 53, and the attendance was 357 members and guests.

The 111th Meeting was held in San Francisco, Cal., Sept. 16 to 19, 1915. Sixty-eight technical papers were presented, and the attendance was over 400 members and guests. The full report of this meeting is given in the *Bulletin* for December, 1915.

Local Section.—A new, strong Local Section of the Institute, with headquarters in Arizona and known as the Arizona Section, was formed during 1915.

Affiliated Student Societies.—Three new Affiliated Student Societies were recognized in 1915 consisting of the Pick and Hammer Club of the University of Oklahoma, Mining and Metallurgical Club of the Univer-

sity of Toronto, and the Scientific Club of the Texas State School of Mines and Metallurgy.

Publications.—Full account of the publications of the Institute is given in the report of the Library Committee. It is to be especially noted that all papers are now put through the Committee on Papers and Publications and the Editorial Department of the Institute and sent to the printer not more than 30 days from the time of receipt, except in a few cases where unavoidable delays occur. Three volumes of *Transactions* were issued in 1915, namely, Volumes XLVIII, XLIX and L. The last of these was sent to members on September 30. Volume LI has been printed and will be delivered to members as fast as their dues for the year 1916 are paid.

Membership.—Full account of this is given in the report of the Committee on Membership. The membership of the Institute on Jan. 1, 1915, was 4,950. The membership on Dec. 31, 1915, was 5,214.

Honorary Member.—Professor James F. Kemp, Sc. D., LL. D., was elected an Honorary Member. A brief notice of Professor Kemp's career was given in the April *Bulletin*.

Library.—A new library agreement was made with the United Engineering Society and the other Founder Societies whereby there is a closer bond of union between the libraries than formerly, with greater efficiency in operation. The Library Board of the United Engineering Society has also carried on more effectively the work of the Library Research Bureau, under the able Chairmanship of E. Gybbon Spilsbury whereby the benefits of the library are extended to members distant from headquarters. Full details on these points are given in the report of the Library Committee.

Additional Space for Members.—For the special benefit of members outside of New York City additional space has been occupied by the Institute on the ninth floor and two Members' Rooms established, where are provided as complete facilities for the convenience and use of out-of-town members visiting the City as can be thought of, including reading, writing, telephoning, messenger service, and opportunities for conferences.

International Engineering Congress.—The International Engineering Congress was supported by the Institute together with four other National Engineering Societies. The proceedings of this Congress took place in San Francisco, September 20 to 25, 1915, and have been reviewed in full in the technical press.

Naval Consulting Board on Inventions.—On July 19, 1915, the Secretary of the Navy addressed a letter to the President of the American Institute of Mining Engineers asking that the Institute nominate two members of the Board which subsequently became known as the Naval Consulting Board on Inventions.

A meeting of the Executive Committee of the Institute was held at which it was decided that a letter-ballot be taken of the members of the Executive Committee to suggest five members of the Institute for this Board; the five names having the highest number of votes were then to be sent to the Board of Directors with the statement that the Board might vote for any of the names so suggested, or for any other members of the Institute. As a result of these two proceedings Messrs. William L. Saunders, and Benjamin B. Thayer were nominated, and were subsequently appointed by the Secretary of the Navy on the Naval Consulting Board. The subsequent operations of the Board have been detailed by

the daily press and also in the Institute's Bulletins of September, October, and December. President William L. Saunders is Second Vice-Chairman of the Board. All the officers and five additional members of the Board are members of this Institute, thus making nine members of the Institute out of the twenty-three members of the Board. The Institute has also extended to the Naval Consulting Board the privileges of its facilities, both in the way of library services and also the use of the Institute rooms for meetings. This invitation has been accepted in the same cordial spirit in which it was offered and the Naval Consulting Board used the offices of the Institute for some eleven meetings on December 22, as well as for the full meeting of the Board that same evening. The Board also held a committee meeting at Institute headquarters on December 31. On December 22 all members of the Board of Directors of the Institute who were then in the city entertained, as individuals, the Naval Consulting Board at dinner at the Engineers' Club.

Second Pan-American Scientific Congress.—The Institute is coöperating with the Department of State of the United States in carrying on the Second Pan-American Scientific Congress, full details of which have been published in the daily press and notice of which was published in the Institute's *Bulletin* for December, 1915. Mr. Hennen Jennings of the Board of Directors of the Institute is Chairman of the Committee on Mining, Metallurgy, Economic Geology and Applied Chemistry, and also a member of the Executive Committee of the Congress. The President of the Institute was in attendance at the Congress during its first week, namely, from December 27 to January 1, and the Secretary of the Institute was in attendance for the second week.

National Reserve Corps of Engineers.—The Secretaries of four National Engineering Societies invited General Leonard Wood to luncheon at the Engineers' Club on Jan. 26, 1915, at which time General Wood outlined a suggestion for a National Reserve Corps of Engineers. This led to action on the part of several societies of which record is given in the October *Bulletin* of the Institute. A joint committee was formed upon which this Institute is represented by Dr. Henry Sturgis Drinker, and the matter is now being deliberated by Congress.

Coöperation with Other Societies.—Besides the coöperative activities mentioned previously in this report the Institute has coöperated at meetings of the American Institute of Electrical Engineers and the Mining and Metallurgical Society of America. At the latter meeting, which was held in Washington, in December, 1915, delegates of the Institute were instructed to vote on behalf of the Institute in favor of a thorough revision of the mining laws and the establishment of a government commission to investigate and make recommendations for the proposed mining law revision. By authority of the Board of Directors the President of the Institute appointed five members of the Institute to coöperate with similar committees from other bodies in keeping before Congress the desire of the mining profession regarding changes in the mining laws, and the establishment of a permanent commission upon this subject.

The members appointed were the following: James R. Finlay, D. C. Jackling, Hennen Jennings, C. F. Kelley and E. B. Kirby.

The Institute has also been represented at meetings of several other societies and has entertained at its meetings delegates of sister societies.

BRADLEY STOUGHTON, *Secretary*.

REPORT OF TREASURER, 1915

*Receipts**General Funds:*

Initiation fees	\$ 4,705.00	
Arrears of dues	1,345.79	
Current dues	40,924.78	
Advanced dues	1,613.29	
Sale of binding	10,367.48	
Sale of advertising	7,109.98	
Sale of transactions	4,085.42	
Sale of special editions	553.17	
Sale of bulletins and pamphlets	2,028.22	
Miscellaneous Receipts:		
Returned to general funds from land fund account	146.50	
Interest on investments and deposits	383.29	
Sale of pins and fobs	278.05	
Library insurance cancelled	103.95	
Sale of library books	28.70	
Refund from Madison Affl. Student Society	11.22	
Miscellaneous	396.47	\$74,081.31
Cash balance December 31, 1914		6,316.90

\$80,398.21*Special Funds:*

Total receipts for land fund account	\$10,646.50	
Returned to general funds of the institute	146.50	
	10,500.00	
Interest on Hadfield Prize	84.70	
Interest on Thayer Prize	1.52	
Life memberships to be invested	1,050.00	\$11,636.22
Cash balance, December 31, 1914		2,181.04
		<u>\$13,817.26</u>

*Payments**General Funds:*

Bulletin	\$15,187.06	
Transactions	11,787.34	
Binding	9,799.41	
Special editions	47.65	
Editorial and office	25,233.63	
Treasurer	753.01	
Library	4,254.75	
Advertising	4,016.30	
Meetings	1,811.21	
Local sections	1,436.62	
Technical committees	267.14	
Back volumes	868.42	
Circulars	455.62	
International Engineering Congress	1,603.01	
Miscellaneous	1,813.65	
December 31, 1915, Cash on hand	1,063.39	
Total		<u>\$80,398.21</u>

Special Funds:

Final payment on land mortgage	\$5,000.00	
Returned to subscribers	5,500.00	
		\$10,500.00
Life membership invested		1,010.56
New York section		240.86
		<u>\$11,751.42</u>
December 31, 1915, Balance on hand		2,065.84
		<u>\$13,817.26</u>

GEORGE C. STONE, *Treasurer.*

REPORT OF LIBRARY COMMITTEE FOR 1915

In accordance with the requirements of our By-Laws I beg leave to submit herewith the report of the Library Committee for the year 1915.

The activity shown by the sale of the Institute publications during the year continues satisfactory, although not as large as last year. The total sales amount to \$6,666.81, distributed as follows:

Sale of Transactions.....	\$4,585.42
Sale of Special Editions.....	553.17
Sale of Pamphlets and Bulletins.....	2,028.22

It should be borne in mind that the sale of special editions was only begun last year, and most of the members who did not have the special editions immediately bought them for their libraries. This year only a little less than half of the same number have been disposed of, and it is more than probable that from now on the number will be considerably lessened.

The additions to the Institute Library during the past year amount to a total of 913 volumes and pamphlets distributed as follows:

	Volumes	Pamphlets	Total
Gifts.....	104	114	218
Exchange.....	557	7	564
Purchase.....	120	2	122
Old material.....	3	6	9
	784	129	913

During the same period the total additions to the combined libraries of the United Engineering Society amounted to 3,214.

The attendance at the Library during the year amounted to 12,790.

During the past year the Library Board has introduced several new features which it is hoped will enable the non-resident members of our Institute to avail themselves to a far greater extent than has heretofore been possible of the Library facilities; this has been done by the establishment of a Library Service Bureau. While this Bureau was only established late last Summer the results have been extremely satisfactory, and tend to prove the necessity there is for such aid to our members.

While before the establishment of this Bureau the amount received for search work as it was then done, amounted on an average to less than \$250 per annum, the past year shows receipts of \$2,666.89, with expenditures to obtain these results of \$2,000.54. Included in the receipts, however, is the sum of \$750 which the three Societies voted to the Research Bureau as a fund for the commencement of their work, so that while the large increase of receipts is satisfactory, the Bureau is still in debt to the Societies for the sum advanced to it.

During the year, the Library has at last succeeded in shelving and cataloguing all of the duplicates belonging to the American Institute of Mining Engineers. Lists of these duplicates—both periodicals and text books—have been published in the *Bulletin* and sent out to the membership at large, with notice that if any of these volumes are desired by members they can be had on application to the Library.

It is suggested by the Library Board of the United Engineering Society, and recommended by your Library Committee, that after the expiration of three months whatever volumes have not been asked for by the mem-

bership shall be disposed of to the best advantage possible either by public or private sale, or failing that, shall be donated to the libraries of Colleges or Institutions, especially those which are connected with our affiliated student branches.

Your Committee further have to report that as members of the Library Board of the United Engineering Society they have attended all the meetings during the year of that Committee.

Respectfully submitted,

E. GYBBON SPILSBURY, *Chairman.*

REPORT OF THE COMMITTEE ON PAPERS AND PUBLICATIONS FOR 1915

During the year 1915 the Committee received 161 manuscripts, which were treated as follows:

- 3 accepted for New York Meeting (110th)
- 64 accepted for San Francisco Meeting (111th)
- 49 accepted for New York Meeting (112th)
- 3 accepted for Arizona Meeting (113th)
- 29 rejected
- 4 returned to authors for revision and not yet accepted
- 9 in committee.

161

Every paper which was presented at the San Francisco Meeting was published in advance in the *Bulletin* with one exception.

The work of the Committee has been systematized and the Executive Committee of the Committee on Papers and Publications has given prompt attention to all manuscripts, with the result that every paper received by the Committee on Papers and Publications is in the hands of the printer within 30 days thereafter, unless delays occur owing to questions as to acceptability, in obtaining from the author permission for revision, or other like causes.

Based upon 53 papers received subsequent to February, 1915, the average time occupied by papers in the different departments is as follows:

	Days
In Committee on Papers and Publications.....	14
In Editorial Department.....	18
Composition, correction, and furnishing of proof.....	9
Transmission to author and return, correcting proof, and final printing.....	22
Total.....	63

While the time in the Editorial Department given above may seem somewhat out of proportion, the Committee reminds the readers of this report that the *Bulletin* is only published once a month; thus some papers are unavoidably held in the Editorial Department for several days no matter how rapidly editorial attention is given to them; other papers require extensive editorial revision, involving correspondence with authors, etc., and these incidents make the average time longer than it otherwise

would be. The Committee also reminds readers of the report that the Editorial Department and the Committee on Papers and Publications are occasionally under a great pressure of papers, especially just previous to a meeting of the Institute, with the result that papers have to wait their turn before receiving attention.

BRADLEY STOUGHTON, *Chairman.*

TEXT BOOKS FOR SALE TO COMPLETE MEMBERS' SETS

In the January *Bulletin* a list was published of Societies' publications, magazines, etc., which were duplicates and were discarded at the time of the consolidation of the libraries of the three Founder Societies. A second list published herewith includes text books which are duplicates and which will be disposed of about April 1, 1916, by the Board of Directors of the Institute through the Library Committee. If, meanwhile, members desire to secure some of these books, they are asked to communicate at once with the Secretary as this is an unusual opportunity to secure books at an extremely low price.

- Airy, George Biddell.—*Magnetism*, London, 1870, Bd.
 Alberti, Friedrich von.—*Halurgische geologie*. Erster Band. Zweiter Band. Stuttgart, 1852, Bd.
 American Iron and Steel Association.—Ann. Rep. of the Sec'y. Statistics of American and Foreign Iron Trades, 1877. Phila., 1877, Bd. Same, for year 1878. Phila., 1879, Bd.
 American Telephone and Telegraph Co.—Ann. Rep. of Directors to Stockholders, 1911. N. Y., 1912, paper.
 Armagnat, H.—*Induction Coils*. Trans. by Otis Allen Kenyon, N. Y., 1908, Bd.
 Armstrong Cork Co. Insulation Dept. Nonpareil Corkboard Insulation. Pittsburgh, 1909, Bd.
 Associazione fra gli Industriali Metallurgici Italiani.—*The Metallurgie Industry of Italy*. Description of some of the principal works. Milan, 1911, Bd.
 Austin, Leonard S.—*Metallurgy of the Common Metals*. (Gold, silver, iron, copper, lead, zinc.) 1st ed., San Fran., 1907, Bd.
 Ayrton, W. E.—*Practical Electricity*. N. Y., 1887, Bd.
 Badt, F. B. and Carhart, H. S.—*Derivation of Practical Electrical Units*. 1st ed., Chic., 1890, Bd.
 Barrett, J. P.—*Electricity at the Columbian Exposition, with Account of Exhibits*. Chic., 1894, Bd.
 Barry, John Wolfe.—*Railway Appliances*. N. Y., 1876, Bd.
 Belgium.—Ministère de l'industrie et du travail. Administration des mines. Watteyne, Victor and Breyre, Adolphe. Les accidents dus à l'emploi des explosifs, dans les mines et carrières souterraines de Belgique, pendant les 15 dernière années (1893 à 1907 inclus.). (Extrait des Annales de mines de Belgique, tome XIV.) Bruxelles (IX.) 1909, paper.
 Billiter, Jean.—*Die elektrochemischen Verfahren der chemischen Groz-Industrie*. Band I—*Elektrometallurgie wässriger Lösungen*. Halle, a.S., 1909, paper.
 Bischof, Gustav.—*Chemischen und physikalischen Geologie*.
 Erster band..... Bonn, 1863, Bd.
 Zweiter band..... Bonn, 1864, Bd.
 Dritter band..... Bonn, 1866, Bd.
 Supplement-band. Lehrbuch..... Bonn, 1871, Bd.
 Bornemann, K.—*Die binären Metallegierungen*. Halle, a.S., 1909, Bd.
 Bradish, Alvah.—*Memoir of Douglass Houghton, 1st state geologist of Michigan*. Detroit, 1889, Bd.
 Callon, J.—*Lectures on Mining*. Trans. by C. LeNeve Foster, and W. Galloway. Vol. 1. Par., 1876, Bd.

Carnegie Library of Pittsburgh.—Classified Catalogs.

Part 1 Gen. works, phil., religion.....	1912, paper
Part 2 Phil. and religion.....	1903, paper
Part 3 Sociology and philology.....	1904, paper
Part 8 Biography.....	1914, paper
Part 9 Bks. for the blind.....	1914, paper

Carnegie Steel Co.—Shape Book. Pittsburgh, 1911, Bd.

Casson, Herbert N.—Cyrus Hall McCormick; his Life and Work. Chic., 1909, Bd.
History of the Telephone. Chic., 1910, Bd.

Cattell, J. McKeen.—American Men of Science. (Biographical directory.) N. Y., 1906, Bd.

Coquand, H.—Traité de roches—leur origine, leur composition, leur gisement, leurs applications a la géologie et a l'industrie.—Minéraux qui fournissent les métaux utiles. Par., 1857, Bd.

Cotterill, James H.—Applied Mechanics. Theory of Structures and Machines. London, 1884, Bd.

Cumming, Linnaeus.—Theory of Electricity. 2d ed., London, 1879, Bd.

Daggett, Ellsworth, Comp. Russell Process; Its Practical Application and Economic Results. Paper presented to A. I. M. E., Boston meeting, Feb., 1888. Author's ed., Salt Lake City, 1888, Bd.

Dahlstrom, Karl P.—Fireman's Guide. Handbook on Care of Boilers. Trans. from 3d ed., London, 1885, Bd.

Dana, Richard T. and Saunders, W. L.—Rock Drilling. Open Cut Excavation and Submarine Rock Removal. 1st ed., N. Y., 1911, Bd.

Daniell, Alfred.—Principles of Physics. 2d ed., London, 1885, Bd.

Davis, Charles Thomas.—Steam Boiler Incrustation, and Methods for Preventing Corrosion. Wash., 1884, Bd.

Del Mar, Algernon.—Stamp Milling. N. Y., 1912, Bd.

Dodge (F. W.) Co.—Building and Engineering Trades Directory, City of N. Y., 1902-1903. N. Y., 1902, Bd.

Du Moncel, Th. and Gerald, Frank. Electricity as a Motive Power. Trans. by C. J. Wharton. London, 1883, Bd.

Dyck, Walther v.—Georg von Reichenbach. München, 1912, Bd.

Engelhardt, Viktor.—Electrolysis of Water. Trans. by Jos. W. Richards. Easton, Pa., 1904, Bd.

Engineering News, Pub.—Manual of American Water-works. 1st annual issue, 1888. Edited by M. N. Baker. N. Y., 1889, Bd.

Engineering Standards Committee.—Report on Progress of Work from Jan., 1901 to July, 1905. London, 1905, Bd.

Everett, J. D.—Units and Physical Constants. London, 1879, Bd.

Flather, John J.—Dynamometers and the Measurement of Power. 2d ed., N. Y., 1900, Bd.

Foot, A. R.—Economic Value of Electric Light and Power. Cincinnati, O., 1889, Bd.

Foot, W. M., Comp.—Complete Mineral Catalog. 12th ed., Phila., Pa., 1909, Bd.

Freeman, John R.—Safeguarding of Lives in Theatres. Reprinted from A. S. M. E. Transactions. N. Y., 1906, Bd.

Frizell, Joseph P.—Water-power. 1st ed., N. Y., 1901, Bd.

Fulton, Charles Herman.—Manual of Fire Assaying. N. Y., 1907, Bd.

Gerard, Eric.—Mesures Électriques. 2d ed., Par., 1901, Bd.

Gesner, Abraham.—Coal, Petroleum, and Other Distilled Oils. 2nd ed., N. Y., 1865, Bd.

Gibbs, William E.—Lighting by Acetylene. 2nd ed., N. Y., 1899, Bd.

Goldingham, A. H.—Oil Engines. 2d ed., N. Y., 1904, Bd.

Gordon, J. E. H.—Electricity and Magnetism. Vols. 1-2, N. Y., 1880, Bd.

Gore, C.—Electrolytic Separation of Metals. N. Y., 1890, Bd.

Grant-Francis, Col.—Smelting of Copper in the Swansea District of South Wales from the Time of Queen Elizabeth to the Present Day. 2nd ed., London, 1881, Bd.

Guertler, Dr. W.—Metallographie. Erster band: die konstitution. Heft 1-12. Ber., 1909-12, paper.

Haupt, Herman.—Bridge Construction. N. Y., 1853, Bd.

Hering, Carl.—Dynamo-electric Machines. N. Y., 1888, Bd. Recent Progress in Electric Railways. N. Y., 1892, Bd.

- Hitt, Rodney, comp.—Master Car Builders' Assoc. Car Builders' Dictionary, 1906 ed., N. Y., 1906, Bd.
- Hodgkinson, Eaton.—Strength and Other Properties of Cast Iron. London, 1846, Bd.
- Hofman, H. O.—Metallurgy of Lead, and Desilverization of Base Bullion. 4th ed., N. Y., 1894, Bd., poor condition.
- Houston, Edwin J.—Dictionary of Electrical Words, Terms, and Phrases. 3d ed., N. Y., 1894, Bd.
- Houston, Edwin J.—Electricity 100 Years Ago and Today. N. Y., 1894, Bd.
- Hunt, Thomas Sterry.—A New Basis for Chemistry; a Chemical Philosophy. 4th ed., N. Y., 1892, Bd.
- Hutchinson, Rollin W.—Long-distance Electric Power Transmission Hydro-electric Generation of Energy; Its Transformation, Transmission, and Distribution. N. Y., 1907, Bd.
- Iles, Malvern Wells.—Lead-smelting. 1st ed., N. Y., 1902, Bd.
- Ingalls, Walter Renton, ed.—Mineral Industry; Its Statistics, Technology, and Trade During 1908, Vol. XVII, N. Y., 1909, Bd.
- Inst. of Civil Engineers.—Heat in Its Mechanical Applications. Lectures delivered at the session of the Inst., 1883-84. London, 1885, Bd.
- Theory and Practice of Hydro-mechanics. Lectures delivered session 1884-85. London, 1885, Bd.
- James Leffel & Co.—Construction of Mill Dams. Springfield, Ohio, 1874, Bd.
- Japing, Eduard.—Elektrische Kraftübertragung und ihre anwendung in der praxis. Nach dem Tode des Verfassers neu bearbeitet von J. Zacharias. Dritte auflage. Wien, 1891, paper, poor.
- Jeans, Wm. T.—Lives of the Electricians. 1st series. (Tyndall, Wheatstone, & Morse) London, 1887, Bd.
- Jenkin, Fleeming.—Electricity and Magnetism. 5th ed., N. Y., 1880, Bd. 9th ed., N. Y., 1887, Bd.
- Johnson, J. B.—Materials of Construction—Strength of Engineering Materials. 1st ed., N. Y., 1897, Bd.
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COMMITTEE ON NATIONAL RESERVE CORPS OF ENGINEERS

The Joint Committee (consisting of the chairmen of the several society committees), formed under the authority of the five societies, in order to facilitate the carrying out of the organization of an engineer reserve as part of the military forces of the United States, for which work it was appointed, now begs leave to report as follows:

The Committee has been in personal communication with the Secretary of War, the officers of the General Staff, the officers of the War College, and with the Hon. George E. Chamberlain, Chairman of the Senate Committee on Military Affairs, and the Hon. James Hay, Chairman of the House Committee on Military Affairs.

As the result of these interviews there has been incorporated in the draft of the legislation proposed by the Secretary of War and included also in the bills drawn by the committees of the Senate and of the House a provision for the organization of an officers reserve on broad lines, including an engineer reserve. As now drawn these bills provide that an officer shall be commissioned for a period of five years with rank up to and including that of major, after passing such requirements of character and qualifications as shall be prescribed by the President. The bill also provides that a certain amount of duty with pay shall be performed each year by the officers in the reserve.

The Committee is waiting for the Navy Department to formulate its plans for the increase in the naval forces, and as soon as a decision has been reached by that Department and the Committee is in possession of the facts, the Committee will take up the question of the organization of an engineer reserve for the navy, similar to that contemplated for the army.

It is the intention of the Committee to attend any Congressional hearings that may be had on the bill or bills introduced for the creation of an engineer reserve.

Prior to the enactment of the necessary legislation, no enrollment for the proposed reserve can be made.

Wm. Barclay Parsons, *Chairman*. Representing Am. Soc. of Civil Engrs. Henry S. Drinker, Am. Inst. of Mining Engrs. William H. Wiley, Am. Soc. of Mech. Engrs. Bion J. Arnold, Am. Inst. of Elec. Engrs. Ralph D. Mershon, Am. Inst. of Consulting Engrs.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Dec. 10, 1915 to Jan. 10, 1916:

A. R. Fletcher, Palo Alto, Cal.	H. L. Hollis, Chicago, Ill.
R. G. Hall, St. Louis, Mo.	Howland Bancroft, Denver, Colo.
Walter E. Koch, Terlingua, Tex.	G. D. B. Turner, Butte, Mont.
George A. Macready, Searchlight, Nev.	H. A. Wheeler, St. Louis, Mo.
H. W. Edmondson, Concheno, Chih., Mex.	George B. Rodgers, St. Francois, Mo.
Arthur F. Taggart, New Haven, Conn.	Paul S. King, Wilmington, Del.
H. V. Winchell, Minneapolis, Minn.	G. H. Clevenger, Palo Alto, Cal.
Frank A. Ross, Spokane, Wash.	J. W. Beard, Syracuse, N. Y.
W. D. B. Motter, Jr., Benson Mines, N. Y.	Albert De Deken, Louvain, Belgium.
S. Le Fevre, Forest Glen, N. Y.	James H. Smith, Jr., Mount Carmel, Pa.

E. H. Hamilton, formerly manager of the Virginia Smelting Co., West Norfolk, Va., has become consulting metallurgist to the Consolidated Mining and Smelting Co., of Canada, Ltd., Trail, British Columbia.

R. J. Johnston has been appointed superintendent of the zinc smelting plant of the American Steel & Wire Company, at Donora, Pa.

W. R. Rust has resigned the presidency of the company operating the Tacoma smelting plant, which is controlled by the American Smelting & Refining Co.

E. A. Cappelen Smith has been elected a director of the Chile Copper and Chile Exploration companies.

W. E. Tracy, for several years past superintendent of the Liberty Bell Co., Telluride, Colo., has resigned and is now in Plainfield, N. J.

John D. Pope for 11 years general manager of the North Butte Mine, Butte, Mont., has resigned. He is succeeded by **NORMAN BRALEY**.

Oscar Cartlidge has accepted the position of General Superintendent of the Illinois Valley Silica Co., Ottawa, Ill.

Charles A. Gibbons, Jr., who has been for the past three years with the Chili Copper Co., Ltd., has severed his connections with that company and is now at Taunton, Mass.

J. B. White is now with the Eclipse Mining Co. at Marion, Ky.

Louis M. Zach, formerly construction engineer with St. Joseph Lead Co. at Bonne Terre and recently with Anaconda Copper Co. at Anaconda is now with U. S. Smelting & Refining Co. at the Salt Lake City office.

W. S. Clarke has been appointed superintendent of the St. Joseph Lead Co., Leadwood, Mo.

Russell G. Wayland has been promoted from assistant superintendent to chief superintendent of the Alaska Treadwell group. **L. H. Metzger**, formerly with the Goldfield Consolidated, succeeds Mr. Wayland as assistant superintendent.

Philip R. Bradley has become general superintendent of the Alaska Juneau mine, retiring as superintendent of the Treadwell group, with which he will continue to be connected as consulting engineer.

Stuart B. Marshall for 21 years connected with the American Manganese Manufacturing Co., Dunbar, Pa., the last 9 years as manager, has severed his connection with this company and accepted a position

with the Aluminum Co. of America, with headquarters at Badin, N. C. Mr. Marshall will have charge of all properties of this company in the Yadkin River Region, Stanley County, North Carolina.

George I. Adams has been appointed Professor of Mining Geology, at the Government University, Peking, China.

William Auman has changed his address from Lykens, Pa., to care of Susquehanna Coal Co., Shamokin, Pa.

Hallard W. Foester has changed his address from Nampa, Idaho, to care of Lucky Tiger Min. Co., Douglas, Ariz.

R. W. Schultz has successfully completed his research work on the hydro-concentration of the low-grade copper-nickel ores of the Sudbury region for the Mond Nickel Co., Ltd., and is spending some time at his home in Ravenna, O.

W. H. Rettie has accepted a position with a company operating in the Belgian Congo; he sailed Jan. 15 to take up his new work.

James W. Smith recently severed his connection with the Wyman Gordon Company, drop forgers, at Worcester and Cleveland. Mr. Smith was formerly Assistant Manager of Wire Mills, Pittsburgh District, American Steel & Wire Co.

Member wishes to meet fellow-member with a view of joining in expenses of downtown office in New York City. Interests non-conflicting with those of practical or consulting mining engineers. Financial references given and required. B. F. P., Care Institute.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Mine and mill accountant for Mexico. Knowledge of Spanish desirable. Salary \$150 to \$200 gold per month with living quarters. No. 60.

Master mechanic familiar with mine and cyanide plant machinery for Mexico; \$200 gold per month and living quarters. No. 61.

Electro-metallurgist for production of ferro-alloys, salary \$200 to \$400 depending on experience. No. 62.

Man familiar with ferro-tungsten products of electric furnaces to draw up specifications for machine tools, to watch these tools in service and revise specifications accordingly. Salary \$150 to \$200 per month. No. 63.

Engineers under 35 years of age for placer mining work in Belgian Congo; \$1,500 to \$1,700 plus traveling and living expenses, two years' contract. No. 67.

Man to take entire charge of the sales of a steel company making electric alloy steels in the shape of ingots, billets and bars. To the right man an opportunity will be offered for taking a leading position in the management of the company and of participating in its results. No. 68.

At the Engineering Experiment Station, University of Illinois, Urbana, Ill., 14 Research Fellowships are open to graduates of universities and technical schools. These Fellowships carry an annual stipend of \$500 and appointments are for two consecutive collegiate years, at the

end of which time a Master's degree may be granted. Full details may be obtained by addressing The Director, Engineering Experiment Station, University of Illinois, Urbana, Ill.

Shift boss of 50-ton cyanide plant in Central America. Salary \$125 per month, traveling and living expenses on one year's contract. Single man with knowledge of Spanish desired. No. 72.

Chemist wanted by Electric Steel concern in the East. Must be fully acquainted with 'alloy' steel analysis. Send full particulars and references. No. 73.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, also Associated Member, A. S. C. E., graduate engineer, aged 31, married, 10 years' experience in gaseous and dusty coal mines, desires position as superintendent or manager. At present employed as general superintendent in West. Good reference from present employers. No. 276.

Member, aged 33, single, technical education, desires position on staff of consulting and examining engineer or firm. Eight years all round experience in gold and silver mines in the United States, Mexico and South America. Speak Spanish. References. Personal interview anywhere in the United States. No. 277.

Member, technical graduate in Mining, Chemistry and Metallurgy. Post-graduate, Royal School of Mines, Freiberg, Saxony, Germany and Columbia School of Mines. Age 37, 14 years' experience in teaching mining metallurgy, chemistry and assaying. Available now. Willing to go anywhere. No. 278.

Member, also member of Am. Soc. Mech. Eng., technical graduate, age 35, with 12 years' experience as engineer and assistant superintendent in mining, milling smelting, railroad and general construction work in the United States and Mexico, desires position either in engineering or operating end. At present employed. Can arrange for New York interview. No. 279.

COLUMBIA LOCAL SECTION

STANLY A. EASTON, *Chairman*, FREDERIC KEFFER, *Vice-Chairman*,
L. K. ARMSTRONG, *Sec.-Treas.*, P. O. Drawer 2154, Spokane, Wash.

The fifth annual meeting of the Columbia Local Section was held in Wallace, Idaho, November 19-20, 1915, with headquarters at the Samuel Hotel.

The afternoon session scheduled for November 19, was adjourned to give the members an opportunity to hear the testimony in the case between the Alameda and Success mining companies on trial in the Superior Court of Shoshone County. The Success Company offered some excellent mine models and maps and furnished some interesting testimony.

In the evening a technical session was held at which Stanly A. Easton presided in the absence of the Chairman and Vice-Chairman of the Section. The following papers were read:

Progress in Ore Dressing By D. W. Buckby.

Mine Accounting for Small Mines, By J. E. Chapman.

Zinc Smelting and Smelter Rates, By W. E. Henry.

In the discussion of the first paper, M. G. Donk, chemist of the U. S. Bureau of Chemistry, reported upon some interesting research he had been conducting to find a cheap substitute for "pine oil" which is now largely used in the flotation process and which is becoming exceedingly scarce.

On November 20, 75 members and guests visited the mills of the Interstate-Callahan, Tamarack, Success and Marsh companies. Each of these mills represented some special feature in the way of milling practice, three of them using some system of flotation for final treatment. All of the mills are of modern design treating either lead or zinc or both. The United Stores Company served a complimentary luncheon to the visitors.

On the evening of November 20, the banquet was held, at which 65 members and guests were present. Following the banquet the Secretary-Treasurer gave his annual report, which showed the membership roll has suffered somewhat since the last report due to unusual removals and resignations. The present status is indicated in the following figures:

Previous membership ¹	154	
Added during 1915 ²	22	176
Removals.....	21	
Resignations.....	25	
Deaths.....	1	47
Net membership at date.....		129

The annual election of officers was then held and resulted as follows: Chairman, Stanly A. Easton, Kellogg, Idaho.

Vice-Chairman, Frederic Keffer, Greenwood, B. C.

Secretary-Treasurer, L. K. Armstrong, Spokane, Wash.

Prof. D. C. Livingston presented a paper prepared by himself and W. N. Ellis on The Leaching of Lead-Zinc Concentrates and Ores of the Coeur d'Alene District.

Brief talks were made by Messrs. E. Jacobs, Glenville A. Collins, Robert N. Bell, Alfred J. Dunn and J. M. Porter.

A party of seven availed themselves of the invitation to visit the Bunker Hill & Sullivan Mine, taking an early train to Kellogg on Sunday morning, joining the main party in the afternoon and then proceeding to Spokane.

L. K. ARMSTRONG, *Secretary*.

SOUTHERN CALIFORNIA SECTION

Executive Committee

SEELEY W. MUDD, *Chairman*,

C. COLCOCK JONES, *Vice-Chairman*,

FREDERICK J. H. MERRILL, *Secretary-Treasurer*,
631 Higgins Bldg., Los Angeles, Cal.

RALPH ARNOLD,
A. B. CARPENTER,

A. B. W. HODGES,
WILLIAM F. STAUNTON.

A meeting of the Southern California Section was held on Dec. 14, 1915, in a private dining room of the Sierra Madre Club, 20 members and guests being present. In the absence of the Chairman, S. W. Mudd, Vice-

¹ Includes some members who removed prior to 1914.

² Includes applicants to Nov. 15, 1915.

Chairman, C. Colcock Jones presided. After a dinner, most enjoyable in culinary and social features, the meeting was called to order and the Chairman introduced the speaker of the evening, Professor Robert T. Hill, of the U. S. Geological Survey, who gave a most instructive and interesting informal talk on the geology and mines of Southern California. At the close of the talk, the subject was discussed by the Chairman, R. B. Moran, and others, and after much social enjoyment the meeting adjourned, the Secretary having announced that the next Section meeting would be held on February 1, to hear a discussion on Petroleum, its Geology and Technology, by speakers to be duly announced.

F. J. H. MERRILL, *Secretary*.

AFFILIATED STUDENT SOCIETY NOTES

The School of Mines Society of the University of Minnesota has held meetings in November and December. On November 20, H. R. Bischoff (of the class of 1910), who has been in Nicaragua examining mining properties for the Crown Reserve Mining Co., told of his experiences in that country and dwelt particularly upon the methods employed in the examination of large properties. His talk was illustrated. On December 15, John S. Stewart gave an illustrated talk on The Everglades of Florida. Mr. Stewart was a geologist with the U. S. Geological Survey for a number of years and subsequently was drainage engineer for the Department of Agriculture. He spoke of the geology, vegetation, animal life and other features of the Everglades which he had encountered while making a drainage survey of that ground.

ADOLPH DOVRE, *President*.

AFFILIATED STUDENT SOCIETIES

The Scientific Club of the Texas School of Mines and Metallurgy, El Paso, Tex., was recognized as an Affiliated Student Society of the Institute by the Board of Directors at its meeting on Dec. 22, 1915, upon recommendation of the Committee on Junior Members and Affiliated Student Societies. The membership of this club consists of active members, limited to upper classmen, and associated members including the lower classmen. The officers of the club are as follows:

VERE LEASURE, *President*,

LLOYD NELSON, *Vice-President*,

L. L. POMEROY, *Secretary*,

J. C. RONAN, *Treasurer*,

O. P. WALKER, *Sergeant at Arms*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

ACCIDENTS AT METALLURGICAL WORKS IN THE UNITED STATES DURING THE CALENDAR YEARS 1913 and 1914. Bureau of Mines. Technical Paper 124.

EXTRACTION AND RECOVERY OF RADIUM, URANIUM AND VANADIUM FROM CARNOTITE. Bulletin 104, U. S. Bureau of Mines. Washington, 1915.

MINERS' WASH AND CHANGE HOUSES. Bureau of Mines. Technical Paper 116. Washington, 1915.

MINING MANUAL AND MINING YEAR BOOK FOR 1915. By Walter R. Skinner. London, 1915.

NOTES ON THE USE OF LOW-GRADE FUEL IN EUROPE. Bureau of Mines. Technical Paper 123. Washington, 1915.

ELEMENTARY PRACTICAL METALLURGY, FOR TECHNICAL STUDENTS AND OTHERS. By J. H. Stansbie, B. S., London. J. & A. Churchill, London, 1915. Price \$1.40. (Gift of P. Blakiston's Sons & Co., Publishers, Philadelphia, Pa.)

NOTE.—This book was prepared for the use of technical students and others who wish to obtain a practical knowledge of the properties of the common metals, by means of laboratory experiments. The book contains chapters on subjects of general interest such as fuel, refractory materials, mechanical testing; and certain chemical changes, a knowledge of which is essential to a clear understanding of the behavior of the metals in the experimental work to follow. The other chapters deal with the non-ferrous metals and alloys, iron and steel, and gold and silver. In each of these chapters instructions are given for carrying out experiments, accompanied by descriptive matter sufficient to render the practical work intelligible.

Geology and Mineral Production

- ALASKA. THE ELLAMAR DISTRICT, ALASKA. Bull. 605, U. S. Geological Survey. Washington, 1915.
- ALASKA. BROAD PASS REGION, ALASKA. Bull. 605, U. S. Geological Survey. Washington, 1915.
- ALASKA. GEOLOGY AND MINERAL RESOURCES OF KENAI PENINSULA, ALASKA. Bull. 587, U. S. Geological Survey. Washington, 1915.
- CANADA. OIL AND GAS FIELDS OF ONTARIO AND QUEBEC. Canada. Mines Dept. Mem. 81. Ottawa, 1914.
- CANADA. RECORDS OF WELLS DRILLED FOR OIL AND GAS IN ONTARIO. Ontario Bureau of Mines, 1915, Vol. 24, pt. 11. Toronto, 1915. (Gift of Ontario Bureau of Mines.)
- CANADA. ARISAIG-ANTIGONISH DISTRICT, NOVA SCOTIA. Canada Mines Dept. Mem. 60. Ottawa, 1914.
- CANADA. PRODUCTION OF COAL AND COKE IN CANADA, 1914. Ottawa, 1915.
- CANADA. PRODUCTION OF COPPER, GOLD, LEAD, NICKEL, SILVER, ZINC AND OTHER METALS IN CANADA, 1914. Ottawa, 1915.
- CANADA. PRODUCTION OF IRON AND STEEL IN CANADA, 1914. Ottawa, 1915.
- CHILE. CHAIN OF PRICES ACROSS FOUR CENTURIES OF CHILEAN COPPER, GOLD AND SILVER BULLION, AND HOW TO PROFIT BY ITS KNOWLEDGE. By Augustus Knudsen. Santiago de Chile, 1915. (Gift of Author.)
- RHODE ISLAND. RHODE ISLAND COAL. Bull. 615, U. S. Geological Survey. Washington, 1915.
- UNITED STATES. U. S. MINT ANNUAL REPORT OF THE DIRECTOR FOR YEAR ENDED JUNE 30, 1915 ALSO REPORT ON PRODUCTION OF PRECIOUS METALS, 1914. Washington, 1915.

Military Engineering

- ENGINEER FIELD MANUAL. Parts I-IV. Ed. 4. (Professional papers of the Corps of Engineers, U. S. Army, No. 29.) Washington, 1912.
- FIELD SERVICE REGULATIONS UNITED STATES ARMY, 1914. Washington, 1914.
- MILITARY RAILWAYS. By Major W. D. Connor. (Professional Papers No 32, Corps of Engineers, U. S. Army.) Washington, 1910.

General

- CODE OF LIGHTING FACTORIES, MILLS AND OTHER WORK PLACES. New York, 1915. (Gift of Illuminating Engineering Society.)
- MOODY'S MANUAL. Complete list of securities maturing Jan. 1, 1916-Dec. 31, 1917. Arranged chronologically. Vol. I. New York, 1915. (Gift of Moody Manual Co.)
- WELDING AND CUTTING BY THE OXYACETYLENE PROCESS. A text book. Minneapolis, n.d.

Trade Catalogues

- AMERICAN BLOWER Co., Detroit, Mich. Sirocco Service. Vol. 5, No. 4.
- AMERICAN MANGANESE STEEL Co., Chicago, Ill. Bull. 52, The Komata Liner. 20 pp.
- BERGER MANUFACTURING Co., Canton, Ohio. The Tests (Metal Lumber System).
- CHICAGO STEEL TAPE Co., Chicago, Ill. Catalogue of manufactures. 35 pp.

- E. I. DU PONT DE NEMOURS Co., Wilmington, Del.
High Explosives, Manufacture, Handling, Storage and Use. 1915.
Kinds, Grades and Brands. 1915.
- GENERAL ELECTRIC Co., Schenectady, N. Y.
Bull. 42206, Curtis steam turbine generators. 1915.
CR 2271. Self starters for small direct-current motors.
- GOLDEN-ANDERSON VALVE SPECIALTY Co, Pittsburgh, Pa. Automatic double-cushioned non-return or stop and check Angle and Globe Valves.
- INGERSOLL RAND COMPANY, New York, N. Y. Form 4221. Jackhammer Drill. Oct., 1915.
- LESCHEN, A., & SONS ROPE Co., St. Louis, Mo. Leschen's Hercules. Dec., 1915.
- SPARTA IRON WORKS Co., Sparta, Wis. Sludge Bucket. Jan., 1916.
- THOMAS, ARTHUR H., COMPANY, Philadelphia, Pa.
Electric multiple replaceable unit furnaces.
Japanese laboratory porcelain ware.
Nonsol flasks and beakers.
Pyrex flasks and beakers as made by Corning Glass Works.
Squibbs analyzed reagents.
Stokes automatic water stills.
"Thelco" electric drying ovens.
Three new analytical balances.
- WEBSTER MFG. Co., Tiffin, Ohio. Webster Method. Nov., 1915.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Dec. 10, 1915 to Jan. 10, 1916.

ANDERSON, FRANK BASIL, Min. Engr., Sampler, Copper Queen Cons. Min. Co.,
Bisbee, Ariz.
BAILEY, LEWIS NEWTON, Min. Engr. 4227 St. James Place, San Diego, Cal.
BELL, JAMES E., Min. Engr., Care Chile Exploration Co.,
Chuquicamata, Chile, South America.
BONILLAS, YGNACIO SAFFORD, Min. Geol., Copper Queen Cons. Min. Co.,
Bisbee, Ariz.
BRADLEY, GEORGE OLIVER, Mech. Engr. Hobart Bldg., San Francisco, Cal.
CLARK, GEORGE L., Min. Engr. Vipond Mine, Schumacher, Ont., Canada.
CONKLING, RICHARD A., Geol., Roxana Petroleum Co. of Oklahoma, Tulsa, Okla.
GOHRING, W. B., Geol., Supt. of Mines, Calumet and Arizona Min. Co., Warren, Ariz.
GROOS, HENRY, Min. and Cons. Engr. Harlan, Ky.
HAGER, DORSEY, Petroleum Geol. and Engr. 211-12 Lynch Bldg., Tulsa, Okla.
HANSEN, CHARLES A., Met., General Electric Co., 9 Waverly Place, Schenectady, N. Y.
McCONNELL, ROBERT E., Min. Engr., 1208 Hollingsworth Bldg., Los Angeles, Cal.
MACKALLOR, DEWITT CLINTON, Min. Engr. 104 E. 15th St., Joplin, Mo.
MALCOLMSON, JAMES DONOVAN, Purchasing Agt. and Cost-keeper,
The Tigre Min. Co., S. A., Esqueda, Sonora, Mexico, via Douglas, Ariz.
NEWTON, CHARLES WHITING, Supt., Cons. Interstate Callahan Min. Co., Wallace, Ida.
NORSWORTHY, HOWARD R., Min. Engr., Société Internationale Forestière et Minière
du Congo, Tshikapa, Belgian Congo, West Africa.
PICK, ALFRED, Engr. Breitung & Co., Ltd., 11 Pine St., New York, N. Y.
PRICE, HAROLD C., Met. Box 871, Bartlesville, Okla.
RAVICZ, LOUIS, Min. Engr., Care Prof. C. P. Berkey, Columbia Univ.,
New York, N. Y.
SIMKINS, WILLIAM A., Min. Engr. 408 Clay Peters Bldg., Reno, Nev.
SMITH, WALTER CRISPELL, Met. Engr., Supt., Silver Building,
United States Metals Refining Co., Chrome, N. J.
STACK, JAMES R., Chemist. American Smelt. & Refg. Co., Perth Amboy, N. J.
STEWART, MELVILLE BOICOURT, Min. Engr. Pekin, Ill.
STIMMEL, BYRON A., Asst. Met., Cons. Min. & Smelt. Co., Trail, B. C., Canada.
VANDEMOER, JOHN, Draughtsman and Engr. 1108 Mills Bldg., El Paso, Tex.
WATSON, FRANCIS, Genl. Mgr., Nitrate Agencies, Ltd., Iquique, Chile, South America.
ZELAYA, CESAR, Min. Engr., Calle de Claras No. 675, Santiago, Chile, South America.

Associates

MESSER, PAUL. Mgr., Engrg. Dept., American Trading Co., Tokyo, Japan.
WALSH, HOWARD THORNTON, Sales Mgr., Sullivan Machinery Co.,
413 Peoples Gas Bldg., Chicago, Ill.
WANJURA, FELIX R. J., Min. Engr. Koskullskulls Iron Mine, Gellivare, Sweden.

Junior Members

ALLEN, GLENN L., Research Fellow University of Utah, Salt Lake City, Utah.
SHAVER, HERBERT HENRY Box 582, State College, Pa.
WENSLEY, ROGER LYTTLE, Student, Hartley Hall, Columbia Univ., New York, N. Y.
Total Membership, Jan. 10, 1916 5,232

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Dec. 10, 1915 to Jan. 10, 1916, for election as members of the Institute. Their names

are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

John Z. Bayless, Thane, Alaska.

Proposed by Erle V. Daveler, B. B. Nieding, G. T. Jackson.

Born 1890, Baltimore, Md. 1905-09, Baltimore Polytechnic Institute. 1910-11, Lehigh Univ. 1913, Univ. of Cal. 1911, Kelley Press Man, Alaska Treadwell Gold Min. Co. 1911-12, Survey Office, Draughtsman, Underground Surveyor's Helper, Alaska Treadwell Min. Co. 1912, Shift Boss, Alaska Juneau Mine, Experimental Man, Alaska Juneau Gold Min. Co. 1912, away on vacation. 1913, Draughtsman, Eng. Office, Alaska Gastineau Min. Co. 1913 to present date, Experimental Mill, Sampler, Assayer Helper, Cyanide Test Work, Sample Foreman, Milling Dept. of Alaska Gastineau Min. Co.

Present position: Met. Engr., Alaska Gastineau Min. Co.

Clinton Percival Bernard, Tshikapa, Congo Belge, W. Africa.

Proposed by Louis D. Huntton, Edward B. Sturgis, E. Maltby Shipp.

Born 1888, Jamaica, N. Y. 1902, Public Schools, Denver, Colo. and New York, N. Y. 1902-06, High School, Jamaica, N. Y. 1906-09, Sheffield Scientific School, Yale Univ., Ph. B. 1909, Engr., Mangas Development Co., New Mex. 1910, Engr. on Churn Drill Prospecting, Cactus Copper Co., Globe, Ariz. 1911, Field Engr., Hempstead, N. Y. 1912-14, Chemist, Engr., Magma Copper Co. and Lincoln Issner Co., Superior, Ariz. 1914-15, Examination work in California, Arizona and British Columbia. 1915, summer, Prospecting and Geological work in Greenland under Sydney H. Ball.

Present position: Min. Engr., Société Internationale Forestière and Minière du Congo.

Jay A. Carpenter, Tonopah, Nev.

Proposed by Edwin E. Carpenter, George J. Young, John G. Kirchen, John L. Dynan, M. J. Conover.

Born 1883, Beloit, Ia. 1901, Grad., High School, Sioux City, Ia. 1901-02, Univ. of Wisconsin. 1902-04, So. Dakota School of Mines. 1904-05, Univ. of Cal. 1906-07, Univ. of Nev., B. S.; 1911, E. M. 1905-06, North Star Mines Co., Grass Valley, Cal. 1907, Nevada Douglas Copper Co., Yerington, Nev. 1908, Belmont Mill Co., Millers, Nev. 1908-09, Instructor, Mackay School of Mines. 1909-10, Asst. Prof. Met., Mackay School of Mines. 1910-11, Designing and building West End Mill, Tonopah, Nev.

Present position: Mill Supt., West End Cons. Min. Co.

Charles Yancey Clayton, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1891, Hannibal, Mo. 1904-08, Hannibal High School, Hannibal, Mo. 1909-13, Missouri School of Mines and Metallurgy, B. S. 1913-15, Grad., Missouri School of Mines. 1913-15, Instr. in Met., Missouri School of Mines and Metallurgy.

Present position: Asst. Prof. of Metallurgy and Ore Dressing, Missouri School of Mines and Metallurgy.

Julius Milton Cohen, Timmins, Ont., Canada.

Proposed by Maurice W. Summerhayes, C. D. Kaeding, C. H. Poirier.

Born 1886, St. Paul, Minn. Public Schools, St. Paul. Mechanic Arts High School. 1904-12, Univ. of Minn. 1907-08, Miner and Timberman, Crown Reserve Mine, Cobalt. 1909-10, Batteryman and Tableman, Temiskaming Mine, Cobalt. 1911-12, Asst. Mgr., Ross Min. and Mill. Co., Silverton, Colo. 1912-13, Mgr., Graphite, Ltd., St. Remi d'Amherst, Quebec.

Present position: 1913 to date; Asst. Mgr., Porcupine Crown Mines, Ltd.

William Stryker Conner, Rancagua, Chile, So. Amer.

Proposed by H. A. Guess, R. K. Stockwell, Lester E. Grant.

Born 1890, St. Paul, Minn. 1902-04, High School, Belleville, N. J. 1906-07, Mt. Hermon, Mass., Preparatory School. 1907-08, High School, Belleville, N. J. 1908-11, Mech. Engrg., Stevens Institute of Technology. 1912-13, Min. Engrg., Mass. Inst. of Tech. 1911, summer, Gillespie Const. Co. Catskill Aqueduct contracts. Tunnel Erectn. 1911-12, Boston and Montana Red. Dept., Amalgamated Copper Co., Special sampler, Mill test work under M. W. Krejci. 1913-14, Braden Copper Co., mill test work, mill office work, oil flotation operator, mill shift boss helper, mill shift boss. 1914, Engrg. Dept. with Mr. R. K. Stockwell.

Present position: Estimating Engr., Braden Copper Co.

Jerome James Day, Northport, Wash.

Proposed by H. J. Stehli, D. N. Ellis, A. S. Dwight.

Born 1876, Truckee, Cal. Common school education. No technical school training. Worked way up from common miner, etc. 1913, Prest., Tamerack and Custer Cons. Min. Co. 1915, Prest., Northport Smelt. & Ref. Co.

Present position: Director, Pennsylvania Smelt. Co.

Willard Albert Deane, Anaconda, Mont.

Proposed by C. R. Kuzell, Albert E. Wiggin, Louis V. Bender.

Born 1881, St. Louis, Mo. 1898, Grad., Chicago Manual Training School. 1903, Special course in General and Electro Metallurgy, Lehigh Univ. 1904, Grad., Michigan Colle of Mines. 1905, Engr., Black Mt. Min. Co., Sonora, Mex. 1906-08, Cyanide Mill Designer, Real del Monte Min. Co., Pachuca, Mex. 1908-09, Chief Draftsman, Guanajuato Dev. Co., Guanajuato, Mex. 1909-11, Supt. Const. and Cyanide Plant, Compania Placera Gardura y Anexas, Placera del Oro, Gaurero, Mex. 1911-12, Battery man, Goldfield Cons. Min. Co., Goldfield, Nev. 1912-15, Draftsman, Anaconda Copper Co., Anaconda, Mont.

Present position: Flotation Research Dept., Anaconda Copper Co.

Charles L. Denison, New York, N. Y.

Proposed by Walter H. Aldridge, W. A. Bostwick, W. W. Main.

Born, Corning, N. Y., 1866. President of the Buffalo Mines, Ltd., for 10 years. Director, Dome Mines Co., Ltd.

Present position: Capitalist and mine owner.

Philip Orr Harding, Rancagua, Chile, So. Amer.

Proposed by R. K. Stockwell, H. A. Guess, Lester E. Grant.

Born 1881, Portland, Me. 1899, Pratt Institute, Applied Electricity. 1899-1900, Draftsman, Marshall Sanders Co. 1900-03, Draftsman, Taylor Iron and Steel Co. 1903-04, Draftsman, Virginia Electrolytic Co. 1904-05, Draftsman, Tennessee Copper Co. 1905-12, Estimating Engr., Cambria Steel Co. 1912-14, Designing Draftsman, Julian Kennedy. 1914-15, Designing Draftsman, Maryland Steel Co. 1915, Designing Draftsman, H. Hoppers Co.

Present position: Designing Draftsman, Braden Copper Co.

John H. Hieber, Kellogg, Ida.

Proposed by J. McD. Porter, Roy H. Clarke, Charles P. Robbins, L. K. Armstrong.

Born 1884, Wickes, Mont. Studied under father, both mining and milling, practical and technical. Partial course, Butte School Mines. Complete course, Int. Correspondence Schools, Metal Mining. Have held positions as both mine and mill shift boss, foreman and superintendent, Monteruma Lead Co., Santa Barbara, Chih., Mex. Badger Min. Co., Corbin, Mont. Keystone Mines Corp., Blacktail, Ida., etc.

Present position: Genl. Supt., mines and mill, Coer D'Alene Antimony Min. Co.

Robert E. Hobart, Lansford, Pa.

Proposed by R. V. Norris, Edwin Ludlow, W. G. Whildin.

Born 1874, Lansford, Pa. 1889, Grad., Tamaqua High School. 1893, Course in International Correspondence Schools (Mechanical Engineering). 1889-92, Machinist apprentice; 1892-99, Machinist. 1899-1908, Draftsman. 1908-11, Genl. Foreman of shops. 1911-12, Asst. Mech. Supt., Lehigh Coal and Navigation Co.

Present position: 1912 to date; Master Mechanic, Lehigh Coal and Navigation Co.

Frederick W. Horton, Denver, Colo.

Proposed by Van H. Manning, C. B. Dutton, Albert H. Fay.

Born 1883, Ipswich, Mass. 1904, Grad., Mass. Inst. of Tech., B. S., in Min. Engrg. and Metallurgy. 1905, Asst. Instr. of Min. Engrg., Mass. Inst. of Tech. 1906, Co-Mgr., Gold Bluff Min. & Lumber Co., Orick, Cal. 1907-08, Field Engr. for various companies, traveling in Alaska; in charge laboratories U. S. Metal Red. Co., Seattle, Wash. 1909, Junior Engr., Technologic Branch, U. S. Geological Survey. 1910, Asst. Editor, Mineral Industry, Hill Publishing Co., New York.

Present position: 1910 to date; Mineral Technologist, U. S. Bureau of Mines.

Charles Ross Huddle, Ivanhoe, Va.

Proposed by E. Gybbon Spilsbury, E. Fleming L'engle, Charles F. Rand.

Born, Ivanhoe, Va., 1884. 1898, High School, Cranberry, N. C. 1899-1900, College, Bristol, Va. 1908, Correspondence course, Elect. Engrg., 1910, Chemical and 1915, Mining. 1899, Asst. to Min. Engr., Cranberry Iron & Coal Co., Cranberry, N. C. 1904, Mech. Dept., Bertha Mineral Co., Austinville, Va. 1906, Supt. Mech. Dept., Ivanhoe Furnace Co., Ivanhoe, Va.

Present position: 1912 to date; Supt., of Mines & Engr., Ivanhoe Furnace Co.

John Bardell Hunley, Seven Troughs, Nev.

Proposed by D. M. Folsom, C. F. Tolman, Jr., C. M. Vrang.

Born 1890, Oneida, Tenn. 1910-14, Leland Stanford Junior Univ., A. B. in Geography and Mining.

Present position: 1914 to date; Assaying, Mine Surveying and bookkeeping, Seven Troughs Coalition Min. Co.

Andre Prokopievitch Ivanoff, Petrograd, Russia.

Proposed by R. Gilman Brown, D. P. Mitchell, R. Macfee.

Born 1874, South Russia. 1897, Imperial Mining Academy, M. E., Petrograd. 1897-99, Asst., Iron Blast Furnaces, South Russia. 1899-1900, One year's travel and study abroad, and Manager of Thomas Steel Plant, Crimea. 1900-05, Genl. Mgr., Alla Verde Copper Mines and Smelter, Caucasus. Built first water jackets for pyrite smelting in Russia. 1906-08, Mgr., Kyshtim Min. Works. 1909-1910, Director of Copper Smelter, Bogoslovsk. 1910-14, Cons. Engr., Russian Min. Society, Petrograd, and other consulting work.

Present position: Managing Director, Ridder Min. & Trading Co., & Kirgiz Min. & Trading Co.

John R. Kenney, Chicago, Ill.

Proposed by Dibrell P. Hynes, H. L. Hollis, Irving Herr.

Born 1886, Drumbo, Ont. Chicago High Schools. Lewis Institute, Chicago, Ill. 1912, Missouri School of Mines and Metallurgy, B. S. 1912-13, Asst. Engr., Experimental Dept., Steptoe Valley S. & M. Co., McGill, Nev. 1913-14, Mill Foreman, Foreman, Buffalo Mines, Ltd., Cobalt, Ont.

Present position: Tunnel Engr., City of Chicago.

S. P. Kinney, Salt Lake City, Utah.

Proposed by Dorsey A. Lyon, Oliver C. Ralston, Walter Neal.

Born 1891, Salt Lake City, Utah. 1911-15, Univ. of Utah, B. S.

Present position: Secy., Arcane Min. Co.

Hugh Bertram Lee, Timmins, Ont., Canada.

Proposed by Maurice W. Summerhayes, C. D. Kaeding, C. H. Poirier.

Born 1888, Columbus, O. 1912, Ohio State Univ., E. M. 1910, Chem., Gas Inspector and Analyst, Consolidation Coal Co., Fairmont, W. Va. 1912-13, Assayer, Mill Man, Shift Boss and Acting Supt., Deadwood Mill of Cleveland and Weatherhead, Mogollon, N. Mex.

Present position: 1913 to date; Assayer and Refiner, Porcupine Crown Mines, Ltd.

Nicolas Lessig, Semipalatinsk Government, Siberia.

Proposed by R. Gilman Brown, D. P. Mitchell, R. Macfee.

Born 1880, Ural. 1907, Mining Institute, M. E., Petrograd. 1908-14, Gold dredging; in charge of iron mines; and for 5 years Director of Copper Mines, Bogoslovsk, Russia.

Present position: Genl. Mgr., Ridder Min. & Trading Co.

Wesley A. Looney, Pittsburgh, Pa.

Proposed by Arthur F. L. Bell, Raymond F. Bacon, Rosewell H. Johnson.
 Born 1865, Philadelphia, Pa. 1881, Philadelphia High School. 1898, Drexel Institute, Philadelphia. 1881-87, Auditor of Passenger Receivers and Genl. Agents Office, Penn. R. R. Co. 1887-1903, Construction and Mfg. Dept., Atlantic Ref. Co.
 Present position: 1903 to date; Genl. Mgr., Gulf Ref. Co.

Charles Frederick Lunt, Rancagua, Chile, So. Amer.

Proposed by H. C. Bellinger, H. A. Guess, R. K. Stockwell.
 Born 1884, Pelsall, So. Staffordshire, Eng. 1894-1900, Queen Mary's Grammar School, Walsall. 1900-05, Birmingham Technical College of Engineering. 1900-05, In works of Messrs. Tangyes Ltd., Birmingham, on general machine fitting and erecting work. 1905-09, In Malay peninsula for Messrs. Tangyes on erection of tin mine and plantation machinery and commercial work. 1909-13, Supervising erection of freezing and textile machinery in Punta Arenas, Chile.
 Present position: 1913 to date; Draftsman, Braden Copper Co.

Ervin W. McCullough, St. Paul, Minn.

Proposed by W. R. Appleby, Peter Christianson, E. H. Comstock.
 Born 1885, St. Anggar, Ia. 1911, Univ. of Minnesota, Engr. of Mines.
 Present position: 1911 to date; Instructor in Mining, The Univ. of Minnesota.

John Spotts McDowell, Pittsburgh, Pa.

Proposed by Waldemar Lindgren, H. O. Hofman, Carle R. Hayward.
 Born 1885, Pocahontas, Va. 1912 to present, have been a student in the mining engineering department of Massachusetts Inst. of Tech. Will complete course of instruction leading to B. S. degree, Feb. 1, 1916. 1903-12, and during summer months since that time, continuously in employ of Harbison Walker Refractories Co., Pittsburgh, Pa. 1903-08, held various positions on engineering corps. 1908 to present, in charge of engineering corps. The work done has included the necessary engineering work from preliminary stage to completion of plant and equipment for coal and fireclay mines, including geological reconnaissance, prospecting, laying out of railroads, haulage and ventilation systems, etc.

Present position: Student, Mass. Inst. of Tech.

Homer I. Merricks, Globe, Ariz.

Proposed by Guy N. Borge, P. G. Beckett, H. P. Bowen.
 Born 1880, Minneapolis, Minn. 1899, East High School, Minneapolis. 1899-1901, 1904-05, Univ. of Minn., School of Mines. 1907-09, Michigan College of Mines, B. S., Houghton, Mich. 1901, Shopdraftsman, American Bridge Co., Minneapolis, Minn. 1902-04, Electrical Dept., Twin City Rapid Transit Co., Minneapolis, Minn. 1905-07, Hydro-Electrical Engr., Stone & Webster Engrg. Corps, St. Croix Falls, Wis. 1909, Engrg. Dept., Soo Line freight, terminal and tunnel const., St. Paul, Minn. 1910, additional construction, Stone & Webster, St. Croix Falls, Wis. 1910-11, Engr., Live Oak Dev. Co. and Black Warrior Min. Co., Miami, Ariz. 1912-14, with various railroad and governmental surveys.

Present position: Draftsman, Mapping Geology, Old Dominion Copper Min. and Smelt. Co.

Hubert Merryweather, Coquimbo, Chile, So. Amer.

Proposed by W. L. Cumings, E. O'C. Acker, C. A. Buck.
 Born 1882, Cincinnati, O. To 1900, Public Schools, Cincinnati. 1900-04, Mass. Institute of Tech., S. B. 1900-04, Various summer jobs in Northwest. 1904-05, Miner and timberman, Calumet and Hecla. 1906-10, Eng., Mill and Mine Foreman in various places in Mexico. 1910-14, Mine Supt. and Genl. Supt., Juragua Iron Co., Santiago de Cuba.

Present position: Vice-Prest. & Genl. Mgr., Bethlehem Chile Iron Mines Co.

Harry Wheeler Morse, Los Angeles, Cal.

Proposed by L. D. Ricketts, A. G. McGregor, Ira B. Joralemon.
 Born 1873, San Diego, Cal. 1897, Stanford Univ., A. B. 1902, Leipzig (Germany) Ph. D. 1911, Fellow, Amer. Academy of Arts and Sciences. 1909-12, Asst. Prof. Physics, Harvard. 1912-13, Lecturer, Chemistry, Univ. of Cal. 1903-10, Consultant, Gould Storage Battery Co.

Present position: Since 1913, in charge of research and development work, Western Precipitation Co.

Walter Innes Nelson, Thane, Alaska.

Proposed by Erle V. Daveler, B. B. Nieding, G. T. Jackson.

Born 1888, Fields Landing, Cal. 1911, Univ. of Cal., B. S. in Min. Engrg. 1911-12, Miner, Alaska Treadwell Mine. 1912-13, Draughtsman, Foreman, Ready Bullion Experimental Plant. 1913, Prospector. 1913-14, Shifter in Cyanide Plant. Refinery foreman, under W. P. Lass, Supt., Treadwell Cyanide Plant.

Present position: 1914 to date; Refinery-man, Alaska Gastineau Gold Min. Co.

Holman Isaac Pearl, Wakefield, Mich.

Proposed by Howard S. Stebbins, James D. Ireland, Allan C. House.

Born 1888, Roslindale, Mass. 1907-09, 1911-12, Civ. Engrg., Massachusetts Institute of Technology. 1908, Sampler and Surveyor, Northwestern Iron Co., Mayville, Wis. 1910-11, Asst. Engr., American Boston Min. Co., Diorite, Mich. 1912, In charge, Magnetic survey for private parties in Wisconsin.

Present position: 1913 to date; Min. Engr.; Genl. Mgr., Wakefield Iron Co.

Irving Perrine, Oklahoma City, Okla.

Proposed by E. DeGolyer, Charles N. Gould, W. E. Wrather.

Born 1884, Wallkill, N. Y. 1901-03, New Palts State Normal School. 1903-12, Cornell Univ., A. B., A. M., Ph. D. 1907-08, Asst. in Geol., Cornell Univ. 1908-09, Asst. State Geol., Louisiana. 1908-11, Instr. in Geol., Cornell Univ. 1911-15, Prof. of Geol., Univ. of Okla. 1912-14 (summers), Prof. of Geol., Cornell Univ. 1913-14, Geol. to J. R. Armstrong, Oil Interests. 1914-15, Geol. to E. W. Marland, Oil Interests.

Present position: Chief Geol., Pierce Oil Corp., and Pierce Fordyce Oil Assn.

Philip Haile Pipkin, St. Francois, Mo.

Proposed by Waldo H. Comins, Herman Garlich, James A. Caselton.

Born 1883, Farmington, Mo. 1899, Grad., Farmington High School. 1903, B. L. Carleton College, Farmington, Mo. 1903-05, Cadet, U. S. M. A., West Point, N. Y. Instruction in practical military engineering, surveying, plotting and mapping. 1906-07, Asst. to Mine Surveyor, St. Louis Smelt. & Ref. Co., St. Francois, Mo.

Present position: 1907 to date; Mine Surveyor, St. Louis Smelt. & Ref. Co.

C. de Ryck van der Gracht, The Hague, Holland.

Proposed by David T. Day, W. D. Waltman, C. E. Jamison.

Born 1866, Sourabaya, Java. 1879-84, High School, Sourabaya, Java. 1901, Polytechnical College, Delft, Holland (Mineralogy and Geology). 1906-07, Secy., Petroleum Maatschappij Saltcreek, The Hague, Holland. 1906-08, Managing Director, Holland Oil Co. and member of the Board of Directors of the Haagoche Petroleum Maatschappij, both of The Hague.

Present position: 1908 to date; Managing Director, Petroleum Maatschappij Saltcreek.

Theodore F. Schwab, Semipalatinsk, Siberia.

Proposed by R. Gilman Brown, D. P. Mitchell, Robert Macfee, T. J. Jones.

Born 1881, Moscow, Russia. 1896, Gymnasium, Moscow. 1904, Urals Mining School. 1901, Cyanide gold ore in the Urals. 1903, Platinum and gold placer work in the Urals. 1904, Workman in Verch-Issetzk Mines. 1905-06, Mine Boss, Andreviski Mine, Kouznetz district, Siberia. 1907, Prospecting for coal in Khirgeez Steppes. 1909, Supt., copper and coal mines in Turkestan. 1910-11, Boss Foreman, Konukhoff Mine, Kyshtim, Urals, Russia. 1911-14, Mines Supt., Tissoff Mine, Kyshtim.

Present position: Supt. (equivalent to Mines Manager) Ridder Mines, Altai district, Siberia.

William Mathew Sirt, Timmins, Ont., Canada.

Proposed by Maurice W. Summerhayes, C. D. Kaeding, C. H. Poirier.

Born 1884, West Park, O. Case School of Applied Science. 1907, Mine Surveyor, Buffalo Mine, Cobalt, Ont., Canada. 1909, Asst. Mill Supt., St. Louis Smelt. & Ref. Co., St. Louis, Mo. 1911, Mill Supt., Nova Scotia Min. Co., Cobalt, Ont., Canada.

Present position: Mill Supt., Porcupine Crown Mines, Ltd.

Henry Boynton Smith, Rivermines, Mo.

Proposed by William Allen Smith, S. Paul Lindau, Franz Casin.

Born 1892, Freehold, N. J. 1911, Grad., Northampton Mass. High School.
Two years, Columbia Univ., School of Mines. 1915, Engineering Dept., St. Joseph Lead Co.

Present position: Asst. Chemist, Doe Run Lead Co.

Harry Howard Stout, New York, N. Y.

Proposed by David H. Browne, J. A. Church, Jr., John A. Church.

Born 1872, Sacaton, Ariz. 1895, Grad., West Point Military Academy. 1895, Officer, Cavalry, U. S. Army. 1898, Cuba Campaign, Santiago. 1899, Officer, Ordnance Dept., U. S. A. Mfg. nitro cellulose. 1901, Genl. Mgr., Peyton Chem. Co., San Francisco, Cal. 1908, Const. Engr., General Chem. Co., New York.

Present position: 1914 to date; Genl. Supt., Nichols Copper Co.

John Wilfrid Wardell, Spassky Zavod, Siberia.

Proposed by R. Gilman Brown, Walter G. Perkins, C. J. Hall.

Born 1889, Winnipeg, Canada. 1900-03, Secondary School, Stockton-on-Tees, England. 1903-10, Technical Institute, Stockton-on-Tees, England; 12 Diplomas in Engineering Subjects. 1910-13, Grad., International Correspondence Schools, Grad., Metallurgy Course. 1903-10, Engineering Apprentice, Head, Whigtson & Co., Ltd., Stocktown-on-Tees, England. 1910-13, Specialist in Milling Plant design, in Engineering office of above company. 1913-14, Member of Engineering Staff of W. G. Perkins & Co., London, E. C., Eng. 1914, Member, Junior Institute Engineers. 1915, Student, Inst. of Mining and Metallurgy.

Present position: Asst. Constructional Engr., Spassky Copper Mines, Ltd.

John Philip Blaikie Webster, London, E. C., England.

Proposed by D. P. Mitchell, T. J. Jones, R. Gilman Brown.

Born 1882, Llandegai, North Wales. Up to 1903, Woburn (Bedfordshire) Marlowes High School, Hertford. Roan School, Greenwich, and subsequent training with McAuliffe and Davis. 1898, Russian Petroleum Co., Ltd., Baku Russian Petroleum Co., Ltd., Russian Collieries and Railway Co., Ltd., London. 1903, Baku Russian Petroleum Co., Ltd., Baku, Caucasus, Russia. 1906, Assisting Leslie Urquhart, in Russian min. work. 1907, Kyshtim Min. Co., Kyshtim, Russia.

Present position: Mgr. and Secy., Irtysh, Tanalyr Russo-Asiatic and Kyshtim Corporations, Ltd.

William Lindsay Wotherspoon, New York, N. Y.

Proposed by David H. Browne, W. A. Bostwick, W. W. Mein.

Born 1879, Liverpool, England. 1896-1901, Durham College of Science, Newcastle-on-Tyne. Apprenticeship to mechanical engineering, W. Gray & Co., West Hartlepool, England. 1901-03, Draughtsman, W. Doxford & Sons, Sunderland, England. 1903-04, Draughtsman, Babcock & Wilcox, London, England. 1904-05, Designing engineer, Richardson, Westgarth & Co., Middlesbrough, England. 1905-12, Occupied varied positions from designer to Asst. Consulting Mechanical Engineer, H. Eckstein & Co., Johannesburg, So. Africa.

Present position: Consulting, Mechanical and Electrical Engr., The International Nickel Co.

Junior Members

Edward E. Bartlett, Pittsburgh, Pa.

Proposed by Roswell H. Johnson, L. G. Huntley, H. B. Meller.

Born 1894, Eldorado, Kans. 1907-12, Oklahoma Agricultural and Mechanical College, B. S. 1912-14, Oil leasing and oil lands, Mannford Oil & Gas Co., Mannford, Okla. and with H. A. Bartlett, Oil Producer, Sapulpa, Okla.

Present position: Student, Univ. of Pittsburgh.

John Rainey Baush, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, G. A. Roush, Howard Eckfeldt.

Born 1894, Beaver City, Nebr. 1910, Grad., Somerset High School, Somerset, Pa.

Present position: Student in Electrometallurgy, Lehigh Univ.

Frederick H. Hayes, Milwaukee, Wis.

Proposed by Richard S. McCaffery, E. C. Holden, J. M. Longyear, Jr.

Born 1890, Plattsmouth, Nebr. High School education. Courses, International Correspondence Schools, Mechanical and Min. Engrg. 1912-15, Sales Dept., Power and Min. Mach. Co., Cudahy, Wis.

Present position: Student, Univ. of Wis.

Howard Andrew Horner, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1894, Pueblo, Colo. 1908-12, Allegheny High School, Pittsburgh, Pa. 1913-15, Pennsylvania State College, State College, Pa. 1912-13, Carnegie Steel Co., Pittsburgh, Pa. 1913, Martin Hardsocg Co., North Side, Pittsburgh, Pa.

Present position: Student, Missouri School of Mines.

Sheu-Chin Hsu, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, Howard Eckfeldt, G. A. Roush.

Born 1891, Chekiang, China. 1910-13, Chekiang College, Chekiang, China.

Present position: Student, Lehigh Univ.

Pang Chieh Loo, Boston, Mass.

Proposed by W. E. Hopper, F. W. McNair, F. W. Sperr.

Born 1891, Shanghai, China. 1915, Grad., Michigan College of Mines, E. M.

Present position: Post graduate of Mining and Metallurgy in Mass. Inst. of Tech.

Donald MacIsaac, New York, N. Y.

Proposed by Howard Eckfeldt, F. J. MacIsaac, W. L. Saunders.

Born 1895, Chicago, Ill. 1910-12, Lewis Institute, Chicago. 1912-13, Manual Training High School, Brooklyn. 1910, Kelso Tunnel. 1911-13, City tunnels, Catskill Aqueduct. 1913-15, Inspecting New York Subway work.

Present position: Student, Lehigh University.

James Ord, Houghton, Mich.

Proposed by W. E. Hopper, F. W. McNair, F. W. Sperr.

Born 1890, Fort Missoula, Mont. 1911, Grad., McKinley Manual Training School, Washington, D. C. 1911-14, Colorado School of Mines. 1914, Denver Park Department (road surveying).

Present position: Student, Michigan College of Mines.

Edward Moore Robinson, Bethlehem, Pa.

Proposed by Howard Eckfeldt, Henry S. Drinker, Joseph W. Richards.

Born 1884, New York, N. Y. 1912-15, Harvard, B. A. 1908-11, St. Paul's School, Concord, N. H. 1915, Laborer and surveyor, inspection in G. B. Markel Co. mines, Jeddo, Pa.

Present position: Student, Lehigh Univ.

Change of Status—Associate to Member

Raymond Chase Bergen, East Orange, N. J.

Proposed by

Born

1907-11, Lafayette College. 1910-13, Mass. Inst. of Tech., S. B. in Metallurgy. 1913-14, Chemist, Amer. Smelt. & Ref. Co., Perth Amboy, N. J. 1914-15, Chemist, and Asst. Plant Mgr., Roessler and Hasslacher Chemical Co., Perth Amboy, N. J.

Present position: Editorial Dept., Metallurgical and Chemical Engineering.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Dec. 10, 1915 to Jan. 10, 1916. This list, together with the list published in *Bulletin* Nos. 100 to 109, April, 1915 to January, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of Jan. 10, 1916.

ADAMS, ARTHUR K. Mascot Copper Co., Dos Cabezas, Ariz.
ADAMS, JOHN H. Box 116, Winchester, Ky.
ALDRICH, TRUMAN H., JR. 1026 Glen Iris Ave., Birmingham, Ala.
ALLEN, CHESTER ARTHUR, Supt. of Construction, Oswego Milling Co., Oswego, N. Y.
ALLEN, ROY H. Room 204, Miners Bank Bldg., Joplin, Mo.
ABE, ASAKA. Care Fujitagumi, Dojima Kitamachi, Kitaku, Osaka, Japan.
ADAMS, GEORGE I. Prof. of the Government University, Peking, China.
ALLEN, MILTON A. Mgr., Pearsite Co., Inc., Clay City, Ky.

AUMAN, WILLIAM	Care Susquehanna Coal Co., Shamokin, Pa.
BEARD, JOHN W.	712 Cortland Ave., Syracuse, N. Y.
BEELE, HENRY C.	254 Coronado Bldg., Denver, Colo.
BOISE, CHARLES W.	709-5th Ave. North, Jamestown, N. D.
BONILLAS, YGNACIO	Subsecretario de Comunicaciones y Obras Publicas, Mexico City, D. F.
BRADLEY, WILLIAM M.	Box 948, Salt Lake City, Utah.
BRIBACH, OSCAR M.	Foerster and Bribach, Box 304, Ouray, Colo.
BROWN, LOWELL H., Prest.,	Battery Engineering & Construction Co., 17 Battery Pl., New York, N. Y.
BROWNSON, EDGAR E.	516-8th St., North, Great Falls, Mont.
CAHEN, JAMES P., JR.	353 Central Park West, New York, N. Y.
CALLAWAY, S. D.	Care Kusa Spelter Co., Dewar, Okla.
CARNAHAN, GEORGE H.	Continental Rubber Co. of N. Y., 120 Broadway, New York, N. Y.
CARTLIDGE, OSCAR, Genl. Supt.	Illinois Valley Silica Co., Ottawa, Ill.
CHAPMAN, IRVING A.	354 Adelphi St., Brooklyn, N. Y.
CHASE, CHARLES A.	812 Cooper Bldg., Denver, Colo.
CHASE, F. D.	Apartado 27, Monterey, N. L., Mexico.
CHAUVENET, S. H.	Prest., Universal Road Machinery Co., Kingston, N. Y.
CLARK, ARTHUR B.	Box 155, Sheridan, Mont.
COLLINS, GLENNVILLE A.	1811 L. C. Smith Bldg., Seattle, Wash.
CORBIN, J. ROSS.	1729 Master St., Philadelphia, Pa.
COUNSELMAN, THEODORE B.	Box 100, Miami, Ariz.
COUSIN, R., Kurall Tin Fields,	Care Naraguta Post Office, Northern Nigeria, West Africa.
COX, WILLIAM J.	326 Colorado National Bank Bldg., Denver, Colo.
CRAWFORD, H. E.	1026 Citizens National Bank Bldg., Los Angeles, Cal.
CROSTON, JOHN J.	Globe, Ariz.
DUCK, GEORGE F.	944 Clay Ave., Scranton, Pa.
EASTON, HARRY D.	Continental Coal Corporation, Straight Creek, Ky.
EDMONDSON, H. W.	465 W. 159th St., New York, N. Y.
EDWARDS, JOHN R.	326 Hope St., Bristol, R. I.
EMERY, WILLIAM, JR.	Madeira, Hill & Co., Pottsville, Pa.
ENOS, H. C.	Binkley Apartments, Apart. No. 10, El Paso, Texas.
EVANS, GEORGE WATKIN.	901 American Bank Bldg., Seattle, Wash.
FLESCHEUTZ, JOHN C.	740 Gas & Electric Bldg., Denver, Colo.
FLETCHER, A. R.	Field Geol., Southern Pacific R. R. Co., Palo Alto, Cal.
FOESTER, HALLARD W.	Care Lucky Tiger Min. Co., Douglas, Ariz.
FORBES, PAUL R., Care Messrs. Shewan Tomes & Co.,	12 Broadway, New York, N. Y.
FOX, WALTER V.	15 Prospect St., Jamaica, N. Y.
GEHRES, GEORGE W.	27 East Ridge St., Lansford, Pa.
GIBBONS, CHARLES A.	Taunton, Mass.
GILLET, LORENZO M.	63 Wall St., New York, N. Y.
GILMORE, LUTHER ELMER.	508 B. Street, Sparrows Point, Md.
GLASS, FRANK A., Supt., Wilcox Mine,	Canadian Cuyuna Ore Co., Woodrow, Minn.
GRUNOW, WILLIAM R.	Care Copper Queen Cons. Min. Co., Bisbee, Ariz.
GUGGENHEIM, EDMOND A.	120 Broadway, New York, N. Y.
GUITERMAN, KENNETH S.	465 West End Ave., New York, N. Y.
HAMILTON, EDWARD H.	Smeltery, Trail, B. C., Canada.
HAMMER, JOHN G.	81-4th St., Henry Bldg., Portland, Ore.
HARDER, EDMUND C.	Minneapolis Athletic Club, Minneapolis, Minn.
HAVLIN, THOMAS NEWELL.	3927a McPherson Ave., St. Louis, Mo.
HAYWARD, CARLE R.	15 Aspinwall Ave., North Weymouth, Mass.
HELLER, MILTON W.	Care The Eagle Min. & Mill. Co., Gilman, Colo.
HENDERSON, CHARLES W.	409 New P. O. Bldg., Denver, Colo.
HENLEY, A. R.	327 Wheeler Ave., Scranton, Pa.
HIESTER, ARTHUR J.	1340 Garfield St., Denver, Colo.
HIGGINS, A. HOVARD.	307 West 98th St., New York, N. Y.
HIGGINS, EDWIN.	407 Underwood Bldg., San Francisco, Cal.
HINCKLEY, E. R.	Silver City, New Mexico.
HINDSHAW, HENRY H., Supt., Michigan Limestone & Chemical Co.,	Buffalo, N. Y.
HOFFMAN, LLOYD.	The Elwood, Tamagua, Pa.
HOLDEN, ROY J., Geological Dept., Virginia Polytechnic Institute,	Blacksburg, Va.
HOLLITSER, SCOVILL E.	Princeton, Mo.
HUTCHINSON, ROBERT M.	332 Pine St., Steelton, Pa.

- JOHNSON, C. B. Care Mineral Springs Hotel, Alton, Ill.
 JOHNSTON, CHARLES W. 325 Boulevard, Port Norfolk, Va.
 KEE, H. A. Nipissing Min. Co., Ltd., Cobalt, Ont., Canada.
 KEEFFER, FREDERIC 610 Hutton Bldg., Spokane, Wash.
 KELLEY, ARTHUR L. Care American Tungsten Co., Tucson, Ariz.
 KELLOGG, R. M. Wellington, Lyon Co., Nev.
 KERR, D. G. 1102 Center St., Wilkinsburg, Allegheny Co., Pa.
 KICHLINE, FRANK O. Saucon Plant, Bethlehem Steel Co., So. Bethlehem, Pa.
 KIRKCALDY, N. M. Shire Engineer, Young, N. S. W., Australia.
 KNOERTZER, HENRI 56 Rue Balagny, Paris, France.
 LARSE, W. S. Care Nevada Cons. Copper Co., Ruth, Nev..
 LASIER, FREDERICK G. R. F. D. 4, Holly, Mich.
 LATHAM, EVERETT BODINE, Deputy State Oil & Gas Supervisor,
 California State Mining Bureau, San Francisco, Cal.
 LECHATELIER, H. 75 rue Notre Dame des Champs, Paris, France.
 LEE, EDWARD CLARENCE 2407 First National Bank Bldg., Pittsburgh, Pa.
 MCCARRICK, E. 213 N. Carondelet Ave., Los Angeles, Cal.
 MCINTOSH, F. K. Florence, Wis.
 McMILLAN, RUSSELL H., 132 Virginia Ave., Aspinwall, Sharpsburg Sta.,
 Pittsburgh, Pa.
 MACIA, JAMES H. Box 540, Tombstone, Ariz.
 VON MALTITZ, EDMUND Genl. Supt., The Hess Steel Corp., Baltimore, Md.
 MARSHALL, STUART B., Care Aluminum Co. of America, Whitney (Badin),
 Stanly Co., No. Carolina.
 MEANS, ELLISON C. Ashland, Ky.
 MILLER, EMORY T. Old Company's Club, Lansford, Pa.
 MILLWARD, WILLIAM Y. M. C., A. Coraopolis, Pa.
 MIXNER, A. F. 10 Queens Chambers, 4 Dalley St., Sydney, N. S. W., Australia.
 MORGAN, GEORGE H., Care Dr. Walter Harvey Weed, Room 302,
 29 Broadway, New York, N. Y.
 NASH, WILLARD H. 19-19th St., Buffalo, N. Y.
 NEAL, WALTER 932 Wilson Ave., Salt Lake City, Utah.
 NORRIS, R. VAN A., JR. 618 W. 113th St., New York, N. Y.
 PAGE, WILLIAM KINGMAN, Care The Chile Exploration Co., Chuquicamata (via
 Antofagasta) Chile, South America.
 PALMER, CHARLES H., JR. 1203 Hollingsworth Bldg., Los Angeles, Cal.
 PORTER, WILLIAM H. Care J. P. Morgan & Co., 23 Wall St., New York, N. Y.
 POTT, JOHN N. 611 South 3d St., Tacoma, Wash.
 PRESCOTT, BASIL 3705 Hueco St., El Paso, Texas.
 PRICKETT, WILLIAM C. 1145-12th St. North, Birmingham, Ala.
 RAMBO, WILLIAM C. G. 1448 De Kalb St., Norristown, Pa.
 RAMSEY, ELMER R. 820 Cooper Bldg., Denver, Colo.
 RANDALL, D. V. Susquehanna Coal Co., Lykens, Pa.
 READ, THOMAS T. Care New Jersey Zinc Co., 55 Wall St., New York, N. Y.
 REBER, BERTRAM A. Care Reber Lighting & Power Co., Absarokee, Mont.
 RICKARD, BRENT N., American Smelt. & Ref. Co., Apartado 101, Monterrey, Mexico.
 SAUNDERS, T. SKEWES, Min. Engr., Genl. Mgr., Cia. Las Dos Estrellas,
 El Oro, Mexico.
 SCHINDLER, DONALD F. Bachelor House, Palmerton, Pa.
 SCHNEIDER, ALBERT F. Care Engineers' Club, 32 W. 40th St., New York, N. Y.
 SCHULTZ, R. W. Ravenna, Ohio.
 SELKIRE, W. Pinners Hall, Old Broad St., London, E. C., England.
 SENGEE, R. W. Care Garfield Smelt. Works, Garfield, Utah.
 SHAW, S. F. Care San Antonio Light, San Antonio, Texas.
 SMITH, EVERETT W., Assayer, Weedon Min. Co., Ltd., Notre Dame des Anges,
 Portneuf Co., Prov. Quebec, Canada.
 SMITH, JAMES H., JR. 114 West Avenue, Mt. Carmel, Pa.
 STARR, CHARLES C. 227 W. Liberty St., Reno, Nev.
 STEEL, DONALD 411 Kipling St., Palo Alto, Cal.
 STEVENS, ARTHUR W. 606 Park Way Ave., Piedmont, Cal.
 STICKNEY, ALFRED W., Care Kyshtim Min. Co., Nevsky Prospect No. 1,
 Petrograd, Russia.
 SWEENEY, HARRY P. 17 Endicott Ave., West Somerville, Mass.
 THOENEN, J. R. Sault Ste Marie, Mich.
 THOMPSON, LESTER S., Engr., Société Forminière, Djoko Punda, Kasai,
 Congo Belge, West Africa.

THOMSON, HERBERT G.	Shift Boss, Care Nevada Packard Min. Co., Lower Rochester, Nev.
TINSLEY, ROBERT B.	Big Stone Gap, Va.
TREAT, F. H.	101 Belmont Road, East Cleveland, Ohio.
TUSCHKA, OTTO	Care Fundicion No. 2, Monterrey, Mexico.
TURNER, G. D. B.	Prest. Revenue Cons. Gold Mine, Butte, Mont.
TUTTLE, ARTHUR L.	Care Tennessee Copper Co., Copperhill, Tenn.
VADNER, C. S.	22½ West 7th South St., Salt Lake City, Utah.
VAN BARNEVELD, CHARLES E.	Min. Dept., University of California, Berkeley, Cal.
VIVIAN, J. C.	Cueva de la Mora, Valdelamusa, Provincia de Huelva, Spain.
VOORHEIS, EDWARD C.	1404 Humboldt Bank Bldg., San Francisco, Cal.
WAGNER, H. R.	Casilla 49-D, Santiago, Chile, South America.
WALTER, E. W.	Keeler, Cal.
WALTERS, MAURICE B.	Standard Mine 5, Silverton, B. C., Canada.
WELLS, JAMES S. C.	Care R. deC. Greene, 105 Thomas St., Cranford, N. J.
WELSH, H. LLEWELLYN	Care Washoe Smelter, Anaconda Copper Min. Co., Anaconda, Mont.
WETHERED, ROY	Tamarack & Custer Cons. Min. Co., Wallace, Idaho.
WHITE, JAMES B., Supt.	The Eclipse Min. Co., Marion, Ky.
WRIGHT, LOUIS A.	Casilla 112-D, Santiago, Chile, South America.
YAKE, E. E.	22 Beaumont St., Springfield, Mass.
YATSEVITCH, MICHAEL G.	Metallurgical Laboratory, Polytechnical Institute, Kiev, Russia.
ZACH, LOUIS MORRIS	U. S. Smelting, Refining and Mining Co., Salt Lake City, Utah.

CHANGES OF ADDRESS OF MEMBERS NOT YET CONFIRMED

AUSTIN, L. S.	251 W. 2d North St., Salt Lake City, Utah.
BATTIN, WILLIAM FREDERICK	Moorpark, Cal.
BURT, CHARLES S.	1322 Hill St., Ann Arbor, Mich.
CROWLEY, T. IRWIN	16 College Green, Dublin, Ireland.
DALY, M. J.	69 Causeway St., Boston, Mass.
LANGLEY, SETH S.	229 Blatchley Ave., New Haven, Conn.
MANN, WILLIAM SEWARD	423 W. Roberts Bldg., Los Angeles, Cal.
McHUGH, PHILLIP M.	812 Cooper Bldg., Denver, Colo.

ADDRESSES OF MEMBERS WANTED

Name.	Last address of Record, from which Mail has been Returned.
BARNES, BLAKESLEE	Care Arrow Engineering Co., Palmyra, Mo.
BELL, D. A. S.	136 McLaren St., Ottawa, Canada.
BLOW, JOHN J.	172 Rodney St., Brooklyn, N. Y.
BOOTH, EVERETT L.	349 W. 145th St., New York, N. Y.
BOYS, HARRY R.	Care Aurora Cons. Min. Co., Aurora, Nev.
BRENNON, J. C.	Tequisquiapan, Queretaro, Mexico.
BUNKER, CHARLES EMMONS	Promontorio, Estacion Chinacates, Durango, Mexico.
CAMPBELL, W. C.	4 Princess St., Roodeport, Transvaal, So. Africa.
COLE, ROBERT J.	McKay Apts., 7th & Pike Sts., Seattle, Wash.
COOK, PAUL RICHARDSON	4159 Grand Boulevard, Chicago, Ill.
CRARY, CHARLES N.	Kimberly, Nev.
DRAPER, CARL H.	Apartado 77, Guadalajara, Jalisco, Mexico.
EATON, EDWIN R.	Manatawny Bessemer Ore Co., Manatawny, Pa.
GOEDICKE, CARL	Box 535, San Antonio, Texas.
GORDON-FIREBRACE, WILLIAM E.	812 Salisbury House, London, E. C., England.
GUNTHER, G. GODFREY	Stratford, Conn.
HALLORAN, WILL	Box 13, Basin, Mont. P. O. N.
HOBART, EDMUND NORRIS	Clifton, Ariz.
HYDE, JAMES M.	1841 Shattuck Ave., Berkeley, Cal.
JONES, THOMAS J.	Kyshtim Min. Wks., Perm Govt., Russia, via Petrograd.
KIRKLAND, T. C., Cia. Met. y Refinadora del Pacifico, S. A.	Fundicion, Son., Me.
KISHMAN, MAURICE W.	101 E. Masonic Ave., Cripple Creek, Colo.
LE NOIR, FRANK H.	Douglas, Alaska.
McKIM, JOHN W.	532 Dooly Block, Salt Lake City, Utah.

MAINWARING, H. M. C.	Chillagoe, Ltd., Chillagoe, Queensland, Australia.
MILLER, FRANK BARTON	Blair, Emeraldalda Co., Nev.
PAGE, JAMES MADISON	Sunnyside Coal Min. Co., Strong, Colo.
PATCHELL, FRED J.	4516 N. Lincoln St., Chicago, Ill.
PATERSON, ARTHUR W.	1814 11th Ave., Spokane, Wash.
PORTER, ROBERT S.	Care Fortifications, Culebra, Canal Zone.
PRETTMAN, FRANK REMINGTON	Oyster Bay, L. I.
RALPH, EDWARD W.	Boston Ely Min. Co., Kimberly, Nev.
RHODES, WILLIAM B.	Golden, Colo.
RINGLUND, S.	Kelly, N. M.
RODRIGUEZ, JUAN C.	Apartado 87, Saltillo, Coahuila, Mexico.
SALE, ANDREW J.	Giroux Cons. Mines Co., Kimberly, Nev.
SEIBERT, PERCY A.	Hagerstown, Md.
SEMPLE, ROBERT ALEXANDER	380 Lenox Rd., Brooklyn, N. Y.
SMITH, ALEXANDER HENRY	Hermant Bldg., Toronto, Canada.
STANLEY, JAMES	147 Holland Road, Kensington, London, W., England.
STODDART, A. W.	638 Salisbury House, London Hall, E. C., England.
SULLIVAN, WILLARD P.	The Henry Walke Co., Norfolk, Va.
TAYLOR, A. W.	Korean Explor. Co., Chiksan Mines, Chiksan, Korea.
TONG, SING KOW	412 W. 115th St., New York, N. Y.
VAN NESS, W. W.	622 Salibury House, London Wall, London, England.
VAN ZWALUWENBURG A.	University Club, Mexico City, D. F., Mexico.
WENTWORTH, IRVING H.	245 Belden Ave., Harlandale Addition, San Antonio, Texas.
WOLFF, MARK A.	618 Pacific Bldg., Vancouver B. C., Canada.
WRAIGHT, ERNEST A.	63 Wavertree Road, Streatham Hill, London, S. W., England.
YOUNG, CARL DEUEL	Tower City, S. D.

NECROLOGY

Date of Election.	Name.	Date of Decease.
1893	*Davis, Morgan, Jr.	Aug. —, 1915.
1902	*Derby, Orville A.	Nov. 27, 1915.
1897	**Oshima, Rokuro	May 4, 1915.
1902	Scarborough, Francis W.	Dec. 24, 1915.

* Member

** Life member.

BIOGRAPHICAL NOTICE

Francis Winthrop Scarborough died in New York City, Dec. 24, 1915. He was born Sept. 6, 1865, at Cincinnati, Ohio. Thoroughly fitted for the high calling of his choice, fitted by nature for any high calling, he died all too young.

The record of his life is brief, but to those who knew him and loved him, that record would be incomplete, even barren of the real truth, did it not contain some expression of the love that went with him wherever he met his fellowmen.

Francis Winthrop Scarborough was graduated from the Rensselaer Polytechnic Institute in 1888. At the Institute he was a member of the Delta Phi Fraternity.

Soon after graduation he entered the service of The Chesapeake & Ohio Railway Company.

He became Engineer of Bridges and Signals, and when he resigned, in 1908, was Chief Engineer of Maintenance of Way and Structures.

Immediately after his resignation he became actively engaged in the operation of coal mines in the New River region of the Chesapeake & Ohio Railway Co., but later went to Richmond, Virginia, where he practised his profession as a Consulting Engineer.

As an officer of the Chesapeake & Ohio Railway Company he commanded the confidence of the Executives, and conducted his organization with marked ability. Added to his scientific attainments, which were notable, and to his fine sense of organization, which was of a high order, was the unique personal gift of eliciting the complete loyalty, and even the affection, of his subordinates. Tireless, himself, as a worker, he imbued those working with him with a like spirit. Thus he became a power on his road.

Mr. Scarborough was peculiarly fearless in his mental characteristics. Ever reasonable, open to conviction, yet when he saw what seemed to him to be the truth, he maintained his position with the utmost firmness, and with unfailing courtesy. His opinions were sought, and were the guides for men handling important matters. All of this is spoken of the accomplished engineer, the able leader and governor of men, the counsellor in his profession.

We, who knew him as the man, think of him as the large-hearted, generous, loyal and beloved friend. We think of him as the unassuming, open-handed delightful companion, ever speaking of all men only that which was kind, with a soul devoid of malice toward any man.

The last months of his life were clouded by sickness, but even then his spirit never lost its brightness, his courage never failed. As all men should, he met his last earthly summons, and when the message came, made answer without fear.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Tests on Various Electric Motor-Driven Equipment Used in the Preparation of Anthracite Coal

BY H. M. WARREN, A. S. BIESECKER, AND E. J. POWELL, SCRANTON, PA.

(New York Meeting, February, 1916)

In the past, steam engines were used in practically all cases for driving the machinery in and about an anthracite breaker, and hence little or no accurate data were available as to the power requirements for starting and operating the individual machines, which make up the complete equipment. It was impossible to segregate the power necessary to drive each individual machine, first, because the equipment was driven in groups, and second, because it was impracticable to obtain graphic, integrated, or instantaneous records of the horsepower used.

As a result of these conditions, it was found, when individual electric motors were applied to the various machines, that more accurate and complete data were needed in order to provide motors of the proper power and design for the particular services to be rendered.

In order to obtain the desired information, the writers made and collected a number of tests, from time to time, on the individual motor drives in electrically operated breakers, and as these tests have been of great value, it occurred to them that a paper giving the results might be of service to other engineers interested in similar installations. Table I shows the results of these tests.

All the motors tested were of the three-phase, 60-cycle, 440-volt, induction, squirrel-cage type, except a few, which, on account of size and starting conditions, were of the wound-rotor type.

The instruments used in making these tests were: An integrating watt-hour meter, a graphic ammeter, and a graphic voltmeter. These instruments were connected in each motor circuit, and a record of a day's run of 9 hours was obtained from each machine.

From these tests, the running light load, the average all-day load, and the instantaneous peak readings in amperes, volts, and kilowatt input, were obtained. By means of the characteristic curves obtained from these motors at the factory, the power factor, the efficiency, and the horsepower output were calculated for each particular test. Starting tests were made by the brake-arm method, in which the torque in pounds was measured directly on a spring balance.

TABLE I.—Tests of Electric Motors, Etc., Employed in Anthracite Breaker

Item No.	Apparatus	Starting				Average All Day				Peak Load	Remarks
		HP	RPM	Eff.	Watt	HP	RPM	Eff.	Watt		
1	Lump Coal Shakers	20	900	100	85	85	84	14.5	115	7.1	
2	Counter Mud	#1	20	85	75	60	85	14.4	76	8.2	
3	Main	#2	20	72	65	74	84	11.5	64	8.0	
4	Counter Main	#1	20	135	85	80	85	9.4	82	5.9	
5	Poney	#1	20	85	65	70	81	9.3	70	6.2	
6	Washery	#1	20	90	51	70	80	10.3	44	7.2	
7	Washery	#2	20	135	77	78	84	15.2	58	10.8	
8	Washery	#3	20	53	45	44	71	6.4	28	4.3	
9	Washery	#4	20	143	82	87	84	17.4	115	21.0	
10	Tailings	#1	20	81	46	81	85	17.3	75	6.7	
11	Tailings	#2	20	144	82	81	85	17.3	80	10.8	
12	Tailings	#3	20	85	73	74	87	9.3	50	6.0	
13	Main Rells	35	900	726	110	85	84	25.0	120	17.5	
14	Main Rells	#2	20	171	146	87	75	5.3	65	5.0	
15	Main Rells	#3	20	225	192	87	75	5.3	67	5.2	
16	Main Rells	#4	20	126	108	86	97	2.2	11	7.2	
17	Main Rells	#5	20	108	93						
18	Main Rells	#6	20	135	115	63	78	6.1	45	4.3	
19	Main Rells	#7	20	72	82	87	75	18			
20	Main Rells	#8	20	55	87	75	84	5.3	53		
21	Main Rells	#9	20	27	47	53	84	3.3	2.1		
22	Main Rells	#10	20	23	40	53	80	3.3	2.1		
23	Main Rells	#11	20	13	111	45	68	3.2	16	3.2	
24	Main Rells	#12	20	22	125	40	58	3.4	12	3.4	
25	Main Rells	#13	20	54	31	74	82	12.8	94	11.0	
26	Main Rells	#14	20	194	167	82	74	4.5	23	3.2	
27	Main Rells	#15	20	170	45	68	3.2	70	3.2		
28	Main Rells	#16	20	67	57	53	75	4.8	24	4.8	
29	Main Rells	#17	20	135	115	144	115	use			
30	Main Rells	#18	20	90	50	31	73	2.1	42	3.6	
31	Main Rells	#19	20	144	44	77	83	2.1	51	4.2	
32	Main Rells	#20	20	42	74	82	85	4.5	53	6.8	
33	Main Rells	#21	20	58	80	70	78	5.5	80	7.7	
34	Main Rells	#22	20	71	64	32	120	14			
35	Main Rells	#23	20	10	34	76	78	2.1	56	2.9	

* These figures apply only when haulage chain is free from ice. The results are based on trip tests, not all-day tests. Note.—Revolutions per minute of driven units figured at synchronous speed of motor. Voltage at motor, 416. Horsepower ratings indicated with O are for back-gear motors. Key: F.L.T. = Full-load torque; P.F. = power factor; Eff. = efficiency; F. load = full load.

Unusual care was taken in making these tests and in calculating the results; therefore, the writers feel that they are reliable for all practical purposes.

As the characteristic curves on the various machines differ, graphic charts taken under actual operating conditions, the results of the tests, and the mechanical data pertaining to the machine driven, follow in proper order.

Shakers

From Table I, it will be noted that tests were made on shakers of various sizes of the two-, three-, and four-deck types. The eccentric

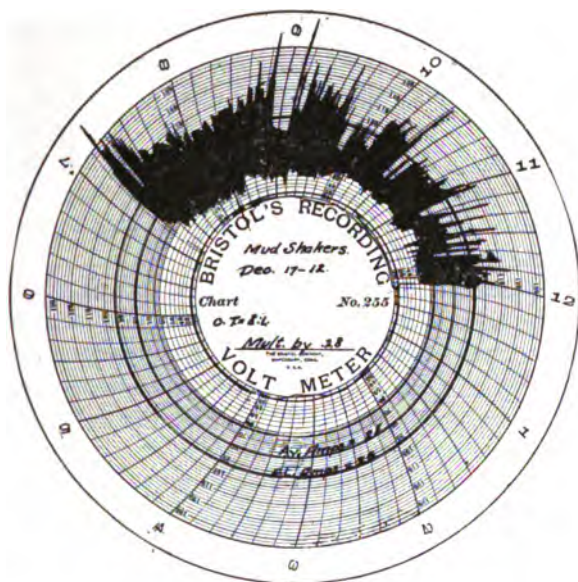


FIG. 1.—GRAPHIC AMMETER CHART SHOWING LOAD ON SHAKER SCREENS.

displacement is divided evenly in order to balance up the load, and the shakers have a 6-in. stroke. There is a slight difference in the speeds at which these various shakers are operated, the speed of the eccentric shafts ranging from 134 to 163 r.p.m.

From the graphic ammeter chart, Fig. 1, it will be noted that the characteristic load on the average shaker screen is a fluctuating one, the fluctuation varying in direct proportion to the degree of unbalancing in the various decks. This unbalancing may be caused by several factors, which are practically impossible to control, viz., unequal quantities of coal on the various decks, unequal weights of the decks, unequal friction in bearings of the suspension rods which support the decks, etc.; also,

a fluctuation in the load may be caused by improper spacing of the eccentrics.

Theoretically, the load on each deck during a cycle of operation conforms very closely to a crank-effort curve as obtained on a steam engine; so, when the load curves of the cycle of operation of a complete shaker are superimposed, the resultant load curve is undulating. This variation in load would tend to show a considerable fluctuation on a graphic ammeter chart where the eccentric spacing is from 180 to 120°, decreasing as the spacing is decreased and the number of decks increased.

The condition which arises in connection with the load obtained on a two-deck shaker when the eccentrics are spaced 180°, deserves special mention. The cycle of operation of the decks when they are driven from the eccentrics spaced 180°, gives the best condition for minimum vibration of the shakers and the maximum variation of load on the motor. If the eccentrics are spaced 90° to obtain the minimum variation of load on the motor, then the maximum vibration is obtained on the shakers. If they were operated in this manner for any length of time, they would set up a very destructive vibration in the surrounding timbers of the breaker. It has been suggested to operate these shakers with 180° spacing on the eccentrics, and install a flywheel on the motor shaft to compensate the load fluctuations.

The writers have had occasion from time to time to investigate troubles occurring in connection with the operation of shakers, and have found in almost all cases that they were due to unequal spacing of the eccentrics at the time of installation; to the attendant's tightening the bearings on the decks or the eccentric straps; to the improper lubrication of the bearings and eccentrics, etc.

Tests and experiments were also made in an endeavor to reduce this fluctuation in load by making use of a flywheel, and also by attaching springs to the various decks; but the results obtained showed that, except possibly in the case of flywheels on two-deck shakers, conditions were not bettered sufficiently to warrant the expense of installing the additional apparatus.

Rolls

The rolls used in the tests and described in Table I were of the manganese-segment type with cast-iron spiders, excepting No. 4 which was of the chilled-tooth type.

The following are the sizes of coal treated by the several pairs of rolls:

Main rolls, Lump to grate coal;

No. 2 rolls, Grate to egg coal;

No. 3 rolls, Grate to egg coal;

No. 4 rolls, In the washery—breaking coal from refuse dump down to egg and below;

No. 5 rolls, In the washery—breaking coal from refuse dump down to chestnut;
 No. 6 rolls, In the washery—breaking coal from the refuse dump from chestnut
 to buckwheat.

The sizes and speeds of these various rolls are found in Table I under "mechanical data."

Since much more trouble has been experienced with the individual motor drive used in connection with main rolls than with any of the other individual drives, the writers have made exhaustive tests and investigations on this particular equipment. Trouble developed with these machines soon after the motors were installed, on account of enormous peaks suddenly thrown on the motor through the crowding of the rolls due to irregularity of feed. This developed into a serious matter, as this

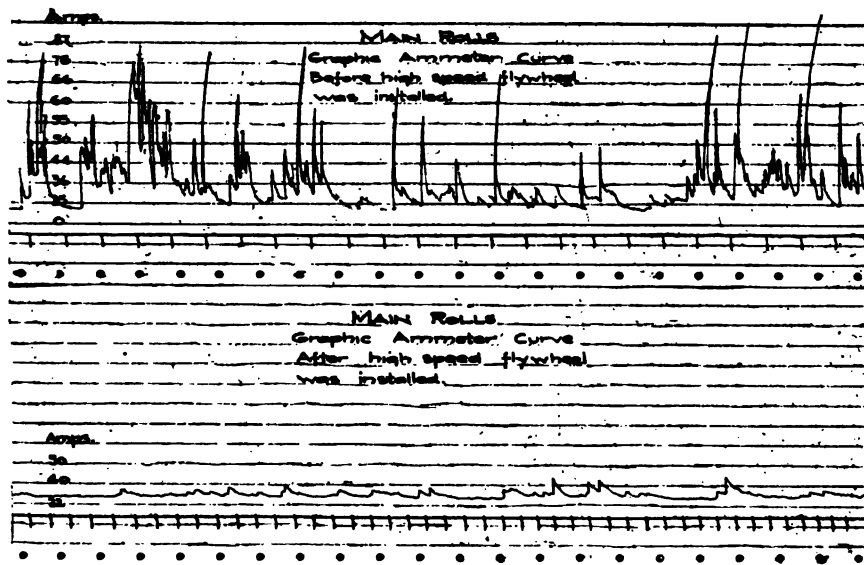


FIG. 2.—GRAPHIC AMMETER CHART SHOWING LOAD ON MAIN ROLLS.

feed reached sufficient proportions to stop the motor, and in time roasted the insulation so badly as to necessitate rewinding. Yet this, in itself, was really a minor matter when compared to the annoyance and the number of delays in the operation of the breaker. In order to locate this trouble, graphic ammeter charts were taken with a special fast-reading instrument, samples of which are shown in Fig. 2.

At this time a 2000-lb. solid cast-iron flywheel was mounted on the roll shaft, and run at about 96 r.p.m. Upon checking up the inertia of this wheel, it was found to be practically valueless at this speed.

From Fig. 2 it will be noted that the load on these rolls varied greatly so that the peaks at times reached the breakdown point on the motor.

In order to rectify this trouble, it was found advisable in some cases

to design a special feeder to deliver the coal to the rolls more uniformly. A type that was found to work well was shaped like a paddle-wheel, and was installed in the chute leading to the rolls. The feeder was revolved by means of a belt driven from a pulley on the motor shaft or from one of the countershafts. Where picking tables or conveyors are used, the feed is sufficiently regular for good operation without the paddle-wheel feeder. After the coal passes through the main rolls, the flow is sufficiently regular on the other rolls for all practical purposes.

In designing special flywheels to take care of the large peak loads thrown on the main rolls, a careful study was made of the graphic ammeter charts obtained from the tests on this equipment. From these peaks it was found that certain conditions had to be taken care of in order to smooth out the curve, and with this information at hand, a special high-speed flywheel was designed and built up from boiler-plate iron.

After this new wheel was installed, other charts were taken with the same high-speed graphic ammeter. One of them is shown in the lower half of Fig. 2. The comparison of the charts shows conclusively that the peak conditions were nicely taken care of by the flywheel. These rolls have not been blocked since it was installed.

The size and capacity of this special wheel were calculated as follows:

By measuring and calculating several peak loads, it was found that the horsepower-seconds of these peaks were about 100. Since the total horsepower-seconds in the old flywheel at 160 r.p.m. was only 95, and for a 10 per cent. slip in the motor there were only 18 hp.-sec. available, and since a watt-hour meter showed that the average load was less than half load on the motor, the inadequacy of the flywheel was evident. It was therefore decided that it would be necessary to replace the old flywheel with a new one so designed that it would carry these peaks without slowing down to such an extent that the motor would be seriously overloaded.

The new flywheel was calculated from the following formula:

$$\text{hp.-sec.} = \frac{WV^2}{2g \times 550}$$

Where W = the weight of the wheel in pounds.

V = the velocity at the radius of gyration in feet per second.

By substituting $\frac{2\pi RS}{60}$ for V we have

$$\frac{WV^2}{2g \times 550} = \frac{W \left(\frac{2\pi RS}{60} \right)^2}{2g \times 550} = \frac{WR^2S^2}{3,200,000}$$

Where R = radius of gyration in feet.

S = speed in revolutions per minute.

In the first place it was decided to design the wheel so that it would give out approximately 160 hp.-sec. for a 10 per cent. slip or reduction in the speed of the motor.

As the horsepower-seconds in a wheel are proportional to the square of the speed, the total horsepower-seconds necessary in order to give out 160 hp.-sec. in slowing down 10 per cent. (or to 90 per cent.) speed would be

$$\frac{160}{1.00 - (0.90)^2} = \frac{160}{0.19} = 840 \text{ hp.-sec.}$$

Since it was desirable to keep the weight of the wheel as small as possible, it was found that the wheel would have to run at the speed of the motor or 900 instead of 160 r.p.m., the speed of the rolls. It was also thought that in order to make the wheel perfectly safe, it would be better to build it up from boiler iron and limit the rim speed to 10,000 ft. per minute. Substituting these values in the above formula, we have

$$840 \text{ hp.-sec.} = \frac{WR^2(900)^2}{3,200,000}$$

As the wheel was to be built up, the radius of gyration would be 0.7 of the total radius. For 900 r.p.m. and 10,000 ft. per minute rim speed, the radius of the wheel is 1.88 ft. or a radius of gyration of 1.31 ft. All of the dimensions of the wheel were thus determined with the exception of the face or thickness. Since a cubic foot of iron weighs about 480 lb., it was found that the thickness of the wheel would have to be 5 in. It is interesting to note that the new wheel weighs only 2,190 lb. or only 190 lb. more than the old wheel, but on account of its speed it delivers nine times the horsepower-seconds in dropping 10 per cent. of its speed.

Another important item for attention is the starting torques of these rolls. Table I shows under the heading of percentage starting torque, that in almost every case over full-load torque is required. The writers suggest for consideration the possibility of using roller or ball bearings in order to decrease the high starting torque.

Conveyors

The conveyor lines are of the ordinary single, chain-flight type, with the exception of the main and the refuse conveyors, which are of the mono-bar and double-chain types, respectively. The length, height, and speed of the line, also the number, area, and the distance between scrapers are shown under "mechanical data" in Table I.

Tests indicate the load to be fairly constant and the friction load very high. A graphic ammeter chart showing the characteristic load on one of these lines is shown in Fig. 3. This chart was taken on the main conveyor line which carries all the coal from the hoisting shafts to the breaker.

There has been no trouble of any importance in connection with the motor driving this line. The size of the motor to be used depends on the elevation and length of the line, the amount of coal to be moved, and possibly on the type of conveyor.

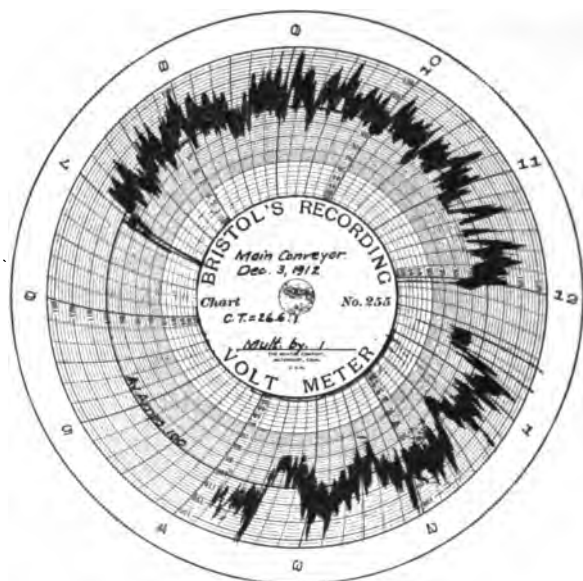


FIG. 3.—GRAPHIC AMMETER CHART SHOWING LOAD ON CONVEYOR LINE.

Elevators

The elevator lines are of the "perfect" and gravity discharge types, equipped with double chains, with the exception of the buckwheat elevator, which has a single chain. The length and speed of the lines, also the number, cubic contents, and spacing of buckets, are given in Table I under "mechanical data."

The load developed on these lines, like that obtained on a conveyor line, is practically constant. A characteristic chart of the load on one of these lines is shown in Fig. 4. This chart was taken on the top elevator which handles the coal from the refuse conveyor and carries it to the top of the washery.

No serious trouble has been experienced with the motors installed on these lines. The determination of the proper size of motor is dependent practically on the amount of coal and the height to which it is lifted.

Jigs

Jigs of the Hazleton standard, single type are operated in groups from a line shaft. Each jig is operated from this line shaft through a counter-

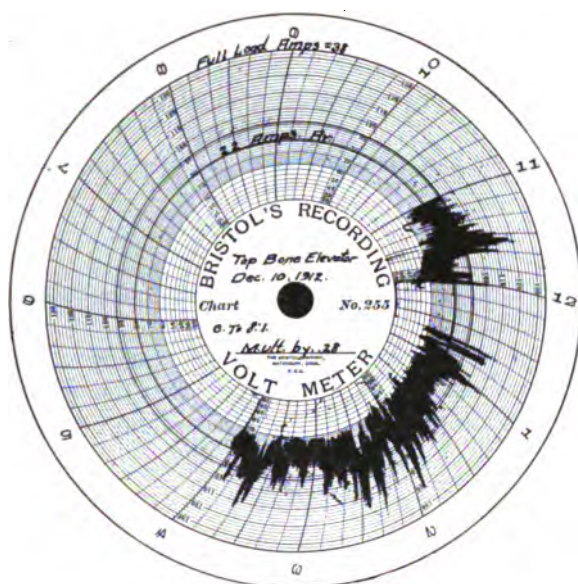


FIG. 4.—GRAPHIC AMMETER CHART SHOWING LOAD ON ELEVATOR LINE.

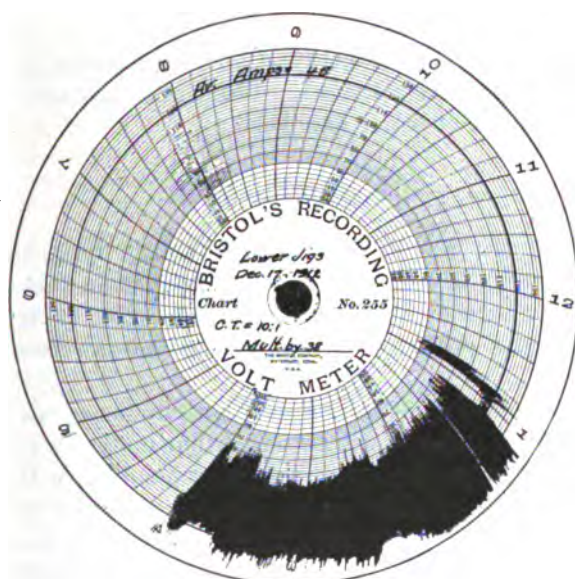


FIG. 5.—GRAPHIC AMMETER CHART SHOWING LOAD ON JIGS.

shaft, on which are mounted the eccentrics necessary for the reciprocating motion required.

From the graphic chart shown in Fig. 5, it will be noted that the characteristic load on these jigs resembles that of the shakers in fluctuating greatly.

No serious trouble has developed with the operation of these jigs.

Dust Fan

The dust fan is of the multivane type and its size, speed, etc., will be found in Table I.

With this type of fan the load is practically constant, depending on the amount of air passing and the speed; it was thought therefore that a graphic record of its performance would be uninteresting.

Car Hauls

Car hauls consist of a single, endless, reinforced chain with 9-in. pitch equipped with knockovers to catch the car. The load is dependent on the running-light friction of the chain, the friction of the cars, the grade of the haul, the weight of the loaded car, and the speed.

For loaded cars or trains there is practically no variation in the power required.

Washery Pump

This pump is of the centrifugal type, and the capacity and head under which it operates are found in Table I. The load is constant, depending on the head and amount of water handled.

Rock Pulverizer

The rock pulverizer is of the bar-and-hammer type, and its capacity is approximately 40 tons per hour. The load on this apparatus is somewhat similar to that obtained on the rolls, viz., being made up of sharp instantaneous peaks, while the average all-day load is comparatively low.

Table II is a collection of tests made on a number of rock pulverizers now in service, from which it will be noted that under average operating conditions 17.8 tons of refuse per hour was handled, with an average consumption of power of 1.488 kw.-hr. per ton, also that the ratio of the average instantaneous peaks to the average load on the motor was 1.92. Fig. 6, a graphic chart taken on one of these rock pulverizers, probably shows the characteristic load to better advantage.

These peaks are due to irregularity in the flow of rock to the pulveri-

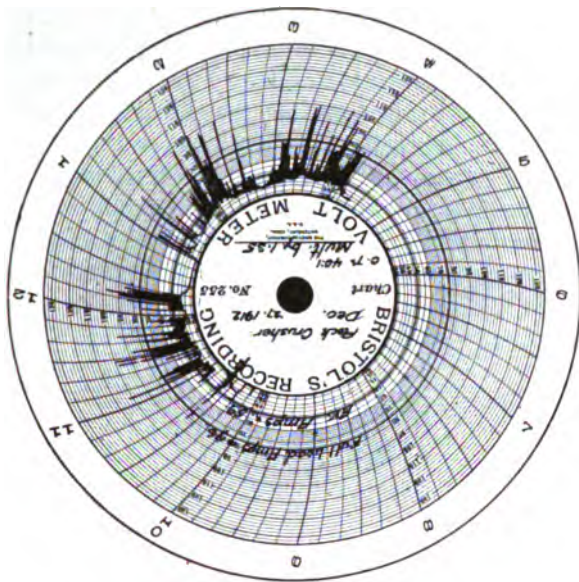


FIG. 6.—GRAPHIC AMMETER CHART SHOWING LOAD ON ROCK CRUSHER.

TABLE II.—Tests on Rock Pulverizers

Colliery	1	2	3	4	5	6	7	Aver.
Motor horsepower.....	75.0	75.0	40.0	75.0	75.0	40.0	75.0	
Motor speed, r.p.m.....	855.0	1110.0	855.0	1110.0	1110.0	855.0	1110.0	
Crusher speed, r.p.m.....	780.0			800.0	850.0	880.0	795.0	
Number of hammers.....	104.0	44.0	44.0	44.0	44.0	44.0	44.0	
Spacing of grates.....	2-in. rd. mesh	0.875	0.875	0.875	0.875	0.875	0.875	
Clearance between grates and hammers, inches.....	1/8 to 1/4	1/4 to 3/4	1/4 to 3/4	1/4 to 3/4	1/4 to 3/4	1/4 to 3/4	1/4 to 3/4	
Tests								
Rock in tons per hour.....	14.0	17.0	10.0	16.0	11.0	20.0	28.0	17.8
Input, average kilowatt.....	31.0	31.2	18.0	25.0	19.7	19.4	28.3	24.7
Output, average horsepower.....	34.0	35.0	19.8	26.0	19.7	19.0	30.0	25.7
Input, kilowatt-hour per ton.....	2.22	1.84	1.8	1.55	1.79	0.95	0.93	1.488
Ratio instantaneous peaks to average load.....	1.6	3.0	4.0	1.55	1.55	2.5	2.4	1.92
Ratio instantaneous peaks to rate load.....	0.84	1.5	3.0	0.75	0.70	2.0	1.25	1.108
Average power factor.....	0.73			0.64	0.55	0.74	0.69	0.648
Load factor.....	0.45	0.46	0.50	0.35	0.26	0.48	0.40	0.388
Fly Wheels								
Diameter in inches.....	None	32.0	32.0	32.0	32.0	32.0	32.0	24.0
Face in inches.....		4.0	4.0	4.0	4.0	4.0	4.0	
Rim depth, inches.....		3.62	3.62	3.62	3.62	3.62	3.62	
Total horsepower seconds.....			213.0	176.0	200.0	213.0	176.0	

zer. The original pulverizers were driven by 50-hp. motors, which were very often loaded to their breakdown point when crowding occurred. In an endeavor to eliminate this difficulty gates were installed in the chutes leading to the pulverizer, so that the attendant could regulate the feed. The writers would not care to recommend this scheme, unless it is in charge of an experienced man, because disastrous results have followed when this machine was operated by a green man. It has been proposed to use a paddle-wheel feeder similar to that mentioned under rolls.

Conveyor lines are sometimes used to carry the rock to these pulveri-

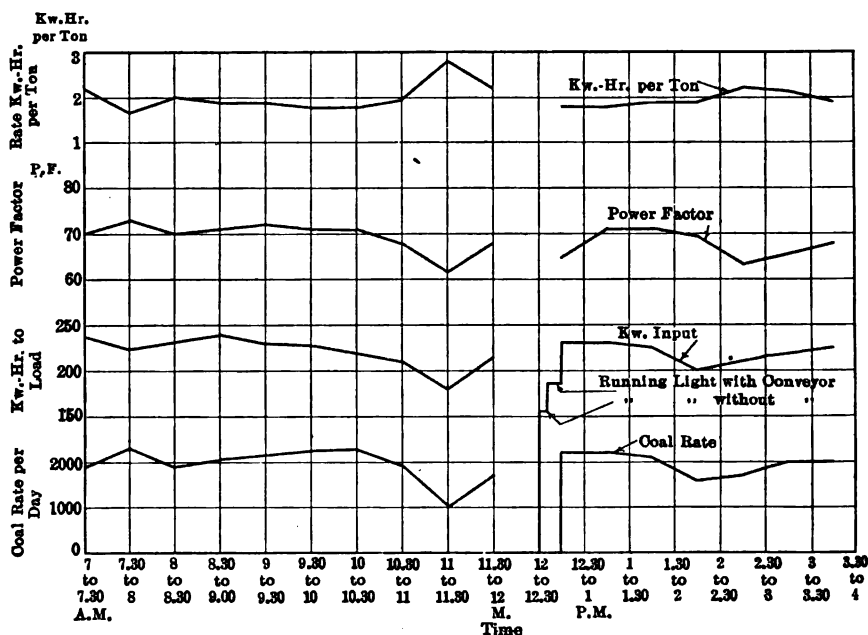


FIG. 7.—CHART OF TOTAL LOAD ON BREAKER, INCLUDING MAIN CONVEYOR, ROCK PULVERIZER, WASHERY PUMP, AND CAR HAUL.

zers, and they act as good feeders. No trouble has been experienced where they were used.

Pickers

The slate pickers are of the standard Emery type, and as the load is comparatively constant and light, and also does not have any inherent characteristic of any interest, no chart is shown of them.

Summary

Curves giving the coal rate, load on the breaker, with and without the main conveyor running, the total kilowatt input, the power factor, and the rate in kilowatt-hours per ton, are shown in Fig. 7.

These curves, as will be noted, are all plotted over time on the base line, and the readings were taken every half hour. It is therefore easy to see how these factors vary with the rate of coal flowing through the breaker.

In Fig. 8, curves are given showing the variations in the kilowatt input, the load and the power factors in the breaker, also the efficiency and the horsepower output of the main conveyor line, with different rates of flow of the coal through the breaker.

Much could be said as to the proper type and design of motors for this service, but as it was thought to be outside the scope of this paper, it was deemed advisable not to discuss them at this time, but simply to say that they should be of heavy and rugged design.

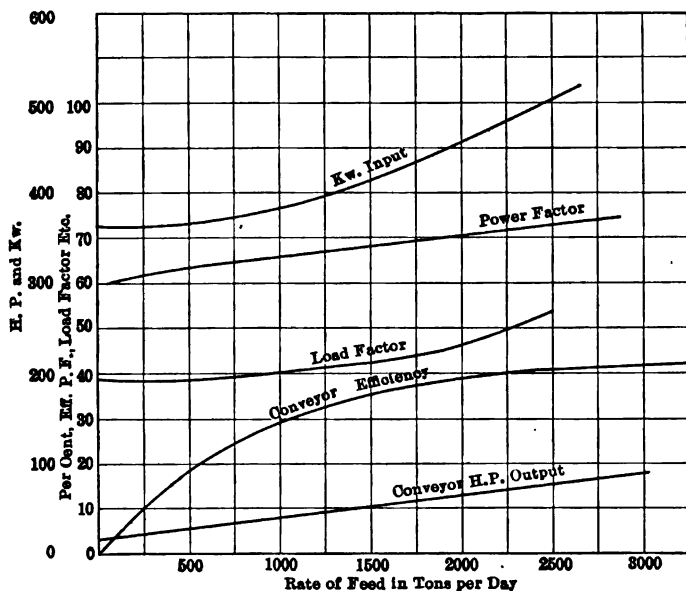


FIG. 8.—VARIATION IN THE KILOWATT INPUT, POWER AND LOAD FACTORS OF THE BREAKER ALSO THE EFFICIENCY AND HORSEPOWER OUTPUT ON THE MAIN CONVEYOR WITH DIFFERENT RATES OF FEED.

In conclusion, the writers wish to emphasize the importance of making a careful study of the characteristics of the machine to which the application of individual motors is to be made, since, for constant loads, the determining point in selecting the size of the machines is dependent on the heating of the motor. Then, again, machines with either high starting torque or large instantaneous peaks, or possibly both, require a motor to meet these conditions rather than for continuous capacity.

To a careful student, some of the motor ratings in the tables here given may seem a little large when compared with the tests; but this is explained by the fact that the breaker on which these tests were made was handling at the time only about 60 per cent. of its rated capacity.

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Interpretation of Assay Curves for Drill Holes*

BY EDWARD H. PERRY,† M. E., CAMBRIDGE, MASS., AND
AUGUSTUS LOCKE,†† M. E., S. D., CAMBRIDGE, MASS.

(New York Meeting, February, 1916)

In the exploration of a copper deposit by drilling, obvious advantages are to be gained from a distinction between primary and secondary ore.¹ Perhaps the chief of these is the aid which such a distinction renders in determining where a given hole should stop. The copper assay, often a sufficient guide, cannot always be exclusively relied upon. Primary ores may extend downward indefinitely and may fluctuate in value independently of depth: if primary ores are consistently lean for a considerable vertical distance, there is little reason to expect that, still deeper, they will increase to the commercial grade; on the other hand, rich primary ores may persist uninterruptedly downward or may come in again below a lean interval. Secondary ores, however, have a comparatively limited vertical extent; in a large way enrichment decreases with depth and finally gives out. It follows then, that when a hole descends from profitable ore into ore of less than the commercial minimum, the practical significance of the situation depends upon whether this decrease results from a decrease in secondary enrichment or is a variation in primary content: if the copper of the ore passed through has been chiefly secondary, little is gained in the average case by extending the hole much below the point at which the commercial limit is reached; but, when copper is chiefly primary, due weight must be given to the possibility that, still deeper, the ore may again improve.

The distinction between primary and secondary ore may of course be effected through an examination of the drillings. It has been found, however, to be facilitated by curves made by plotting assays against

* This is one of a series of contributions by the Secondary Enrichment Investigation that are intended to present results of specific portions of its work in advance of the complete memoir.

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¹ In this paper, the terms *secondary* and *enriched* are used in the sense commonly ascribed to them in the expression "secondary enrichment," and refer to changes undergone by initial, i.e., *primary*, constituents of an ore as the result of solutions descending from the surface. The term *ore* as here used does not necessarily signify material of commercial grade and consideration is given only to sulphide, not to oxidized ores.

depths. In this connection, the data of several hundred drill holes in the copper districts of Ajo, Bingham, Ely, Miami, Ray, and Santa Rita, courteously furnished by officials of the local companies, have been studied in conjunction with the corresponding drill pulps and ore specimens. The accompanying curves represent holes chosen to illustrate both normal and abnormal conditions, which are respectively simple and difficult of interpretation; they are intended particularly to indicate how a presentation of the facts by their means readily permits deductions concerning secondary enrichment in copper ores.

Normal Conditions in Porphyry² and Schist

The ores, whether primary or secondary, that are best adapted to exploration by drilling are those in which the valuable minerals are uniformly distributed. The result of enrichment on primary material of this kind is an ore not only richer but of somewhat more variable grade, though still sufficiently uniform to be suitable for drilling. Since a condition of uniform distribution is that most commonly met in drilled deposits, it may for the purposes of this paper be regarded as normal. Though present in some other rocks, it is most likely to occur in porphyry or schist.

The significant features of curve-shape are chiefly concerned with the abruptness of the change in copper content. In particular, a jagged profile is to be contrasted with a smoother one, and, according as they possess one or the other of these characters, curves representing normal conditions fall into two principal groups:

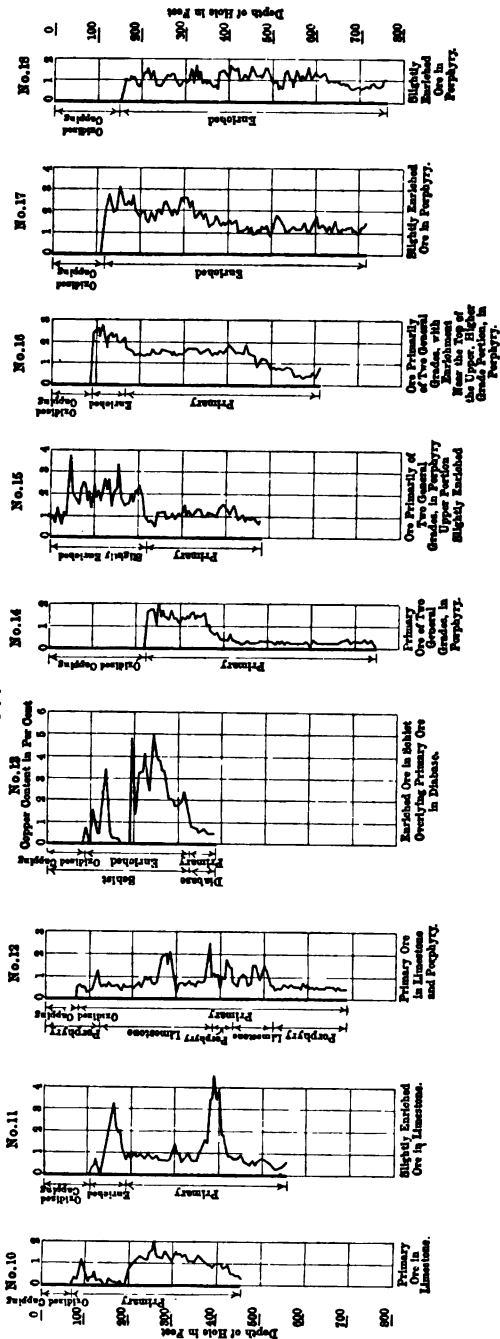
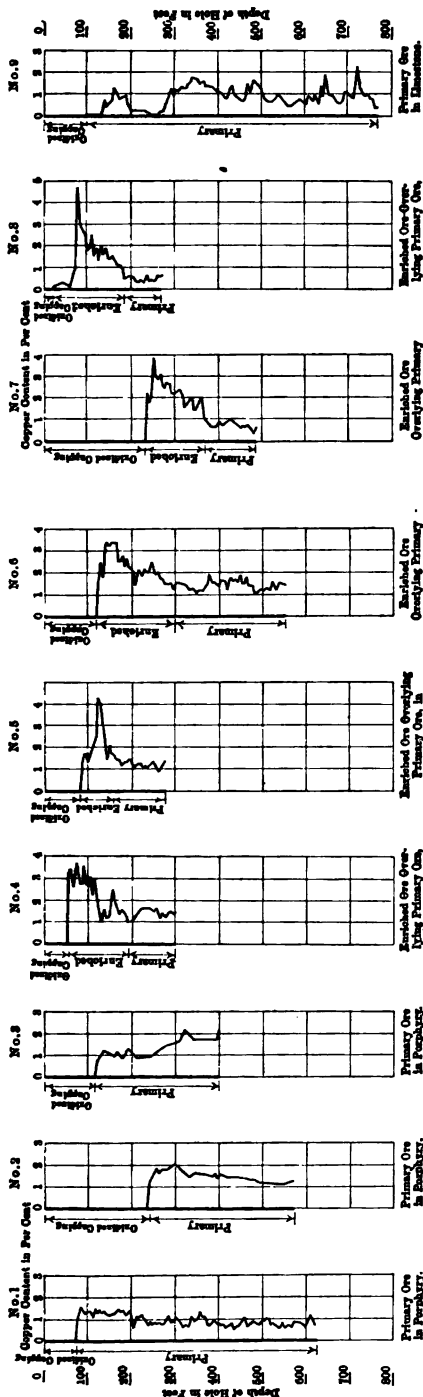
I. That in which the tenor undergoes no abrupt change throughout the sulphide part of the hole (*e.g.*, curves 1, 2, and 3).

II. That in which the lower part of the curve has the characteristics of group I, while the upper part is richer and is marked by peaks (*e.g.*, curves 4 to 8).

Primary ore commonly yields a curve of mild profile. A curve of secondary ore is commonly peaked, and, as might be expected, the more emphatic the enrichment, the higher and rougher are the peaks, and the more characteristic is the form of the curve. A curve, then, of group I presumably means primary ore, while a curve of group II presumably means primary ore below secondary ore. The curves noted as illustrating these two groups are, in fact, exactly what their forms thus suggest.

In group I, curves 1 and 2 well exemplify, by the smoothness of their profiles, the comparatively uniform tenor of primary ores under conditions defined as normal. They also illustrate a not unusual tendency of primary ore to decrease gradually in value with depth. Curve 3 indicates a somewhat less constant grade of ore, yet sufficiently uniform to

² The term *porphyry* is used in the popular and not in the strictly technical sense.



place the curve in group I; in any case, there is nothing in its shape to suggest enrichment.

In group II, curves 7 and 8 bring out strikingly the effect of strong enrichment on low-grade primary stuff; in curves 4 and 5, though the primary ore was of better grade and the ratio of enrichment somewhat lower, the effect of enrichment is clearly shown by the distinct difference in character as well as in tenor between the upper and lower parts of the curves; in curve 6, good primary ore, not very strongly enriched, yields a profile less strikingly different in its two parts, though even here the situation is plain. In all these five curves, the conditions are to be regarded as normal.

It is fortunate that these two types of curves, which are generally easiest of interpretation, are the ones most commonly encountered.

Disturbing Influences of Other Rocks

An interpretation of ore character, however, by means of curve shapes alone is often beset with difficulties. Departures from normal conditions may be encountered in porphyry or schist but are met more frequently in other rocks. They are most likely to be found in limestone, which is fairly common in deposits explored by drilling, and which, on account of the erratic nature of its mineralization, often introduces disturbing factors. Various kinds of difficulties that are experienced find illustration in curves 9 to 13, which may be classed as still another group:

III. That in which decided irregularities occur once or more, without systematic relation either to each other or to the top of the hole.

Curves 9 and 10, representatives of a type which is common, are of primary ore in limestone. If they were in schist or porphyry their moderately peaked character extending to the bottom would suggest deep and rather mild enrichment.

Curve 9 serves to illustrate also how the copper assay alone might lead to an erroneous decision as to the depth at which the hole should be bottomed: on the basis of copper assay alone, the hole might have been stopped at a depth of about 250 ft. or at a depth of about 600 ft.; if the decline in tenor just above either of these horizons had been due to a playing out of enrichment, bottoming at one or the other of these depths would have been justified; but since the ore is in fact all primary, it was wise to sink the hole deeper and actually there is no indication that the hole is yet deep enough.

Curve 11 is of slightly enriched ore in limestone. The existence of two peaks [widely separated is not typical of enrichment in schist or porphyry, and suggests primary ore of erratic or bunched distribution and this in turn suggests limestone. The two peaks in the curve might, for example, coincide with two of the limestone beds that were

especially favorable for primary-ore deposition. As a matter of fact, were the effect of enrichment removed from this curve, its shape would be only slightly changed. This situation well illustrates the impossibility of detecting enrichment in limestone from the shape of the curve alone.

Curve 12 is of primary ore in alternating porphyry and limestone. It has the appearance of being a typical curve of primary ore in limestone, but there is no way of determining from its shape whether or not enrichment is involved in any part of it.

Curve 13 is of enriched ore in schist overlying primary ore in diabase. Altogether, the curve looks like one in porphyry or schist with irregular enrichment above primary ore; the barren stretch near the middle of the hole might be variously interpreted as the result of leaching or as due to primary poverty, and the small length of low grade at the bottom might suggest a return to primary conditions. As a matter of fact, the low grade at the bottom is due to the presence of the diabase into which the enriching solutions were unable to penetrate effectively, and the barren interval is the result of oxidizing influences, which have entered obliquely or laterally, and have thus undercut sulphides lying above.

Erratic Conditions in Porphyry and Schist

Abnormalities which are occasionally present in deposits contained in porphyry or schist may bring about conditions as difficult of interpretation as those due to the presence of limestone or of other disturbing rocks, or they may yield curves which, on the basis of shape, would fall in group II or even in group I, but which in reality owe their shape to causes other than those normally determining the profiles characteristic of groups I and II respectively. Curves 14 to 16 afford examples of such erratic or abnormal conditions.

Curve 14 is of primary ore in porphyry. Viewed broadly, it suggests enriched ore over primary ore; yet its upper and richer part lacks the strongly serrated outline usually found to accompany enrichment. Indeed, each of the two parts, taken independently, looks primary, and this is actually the case, the drop in tenor coinciding with the entrance of a porphyry of a different variety which, in the district in question, carries less copper.

Curve 15 is of ore in porphyry, only slightly enriched, and representing two primary grades, the poorer underlying the richer. The primary copper of the poorer is in chalcopyrite; that of the richer is in bornite as well as chalcopyrite. On the basis of shape, the curve is an especially good example of group II, which would ordinarily signify enriched ore overlying primary ore; yet, in this particular case, the effect of enrichment on the shape of the curve is only to accent the relief of the peaks in its upper, richer part.

Curve 16, of ore in porphyry, might be thought to indicate deep enrichment giving way to primary ore near the bottom. As a matter of fact, enrichment is confined to the upper part, where the irregularities are most accentuated; below the point, at a depth of about 170 ft., where the tenor suddenly steps down and the curve becomes smoother, the ore is wholly primary, growing distinctly leaner near the bottom.

Curves 17 and 18 are in porphyry and are slightly enriched throughout. They are not to be regarded as abnormal; they represent a normal condition which yields a somewhat erratic curve. Nothing in their shape, however, distinguishes them from curves of primary ore in limestone or even perhaps of primary ore of somewhat variable grade in porphyry or schist.

The causes of the lack of primary uniformity encountered in some ores in porphyry and schist are various and often obscure. Low-grade pervasive primary mineralization is in certain cases localized by the contacts between intrusive rocks and the rocks which they invade; vagaries in the distribution of sulphides may therefore be concerned with a neighboring contact. The presence of a mineralized dike, or of a vein, or an unaccountable increase of disseminated primary sulphides, may, for example, produce irregularities that look like the peaks of an enrichment curve. The fact, moreover, that, in the districts involved, enrichment always takes place through the replacement of one sulphide by another, and that it acts selectively, in such a way that bornite yields to its effects more readily than chalcopyrite, and both of these more readily than pyrite (to name only the three primary sulphides most commonly involved), causes peaks due to primary richness to undergo a misleading exaggeration, as is illustrated in curve 15.

Requisites for Correct Interpretation

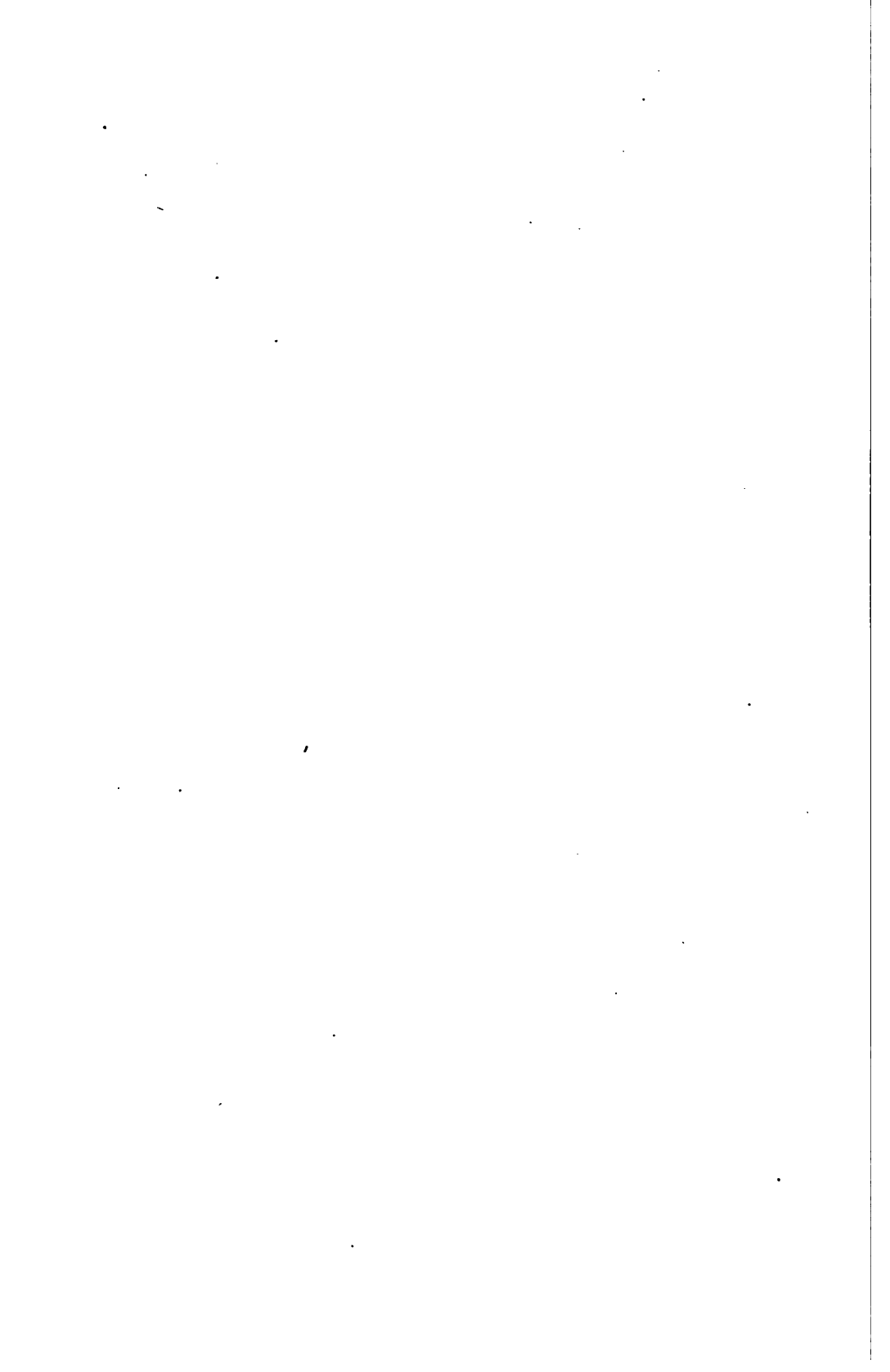
After all, there are a number of varieties of knowledge, not yielded by curves, which are essential to an intelligent diagnosis of conditions. The study of curve shapes must be made with an understanding of the geologic habits of the district. Not uncommonly there is a decrease in the grade of primary ore with depth which may involve an entire district or affect only a certain portion; it may be related to the position of a contact or dependent on some other influence, but whatever its cause, it must be taken into consideration if the curves are to be correctly understood. A knowledge of the kind of rock traversed is invariably required and there must always be enough study of drillings and of other specimens to yield an intimate acquaintance with the district habits.

Such studies are facilitated by the microscope. Thin sections are sometimes necessary for the identification of the rocks and determination of the alterations they have undergone. Polished surfaces of opaque

minerals are often indispensable for the recognition of enrichment and of its character. Powders, such as drill pulps, can be investigated by the microscope, or by the binocular, a magnetized needle aiding in the separation of magnetite from chalcocite which, in the powdered form, it closely resembles. It is often advantageous to embed powders in sealing wax, in order that polished surfaces of their constituent grains may be prepared for microscopic examinations.

Limestone—which is especially important to recognize on account of its disturbing influences—is sometimes difficult of identification in pulp form because of intense alteration. The common test of effervescence with acid may, of course, be ineffectual, because of the frequent destruction of the carbonate by the alteration. A familiarity with the conditions or character of limestone alteration in the district will usually serve in separating this rock from porphyry. The limestone, however much altered, is likely to have a characteristic aspect, due, for example, to certain habits of sulphide aggregation or of softness.

Altogether, the usefulness of a study of curve shapes will vary inversely with the amount of additional knowledge needed for their interpretation. Should the deposit or district be erratic in its habits, curves may not be very intelligible. But should the district possess a uniformity or regularity in its general geology and its mineralization, the curves are likely to be very significant and to constitute an important short-cut to an understanding of the ore distribution.



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An Electro-Hydraulic Shovel

BY FRANK H. ARMSTRONG,* B. S., VULCAN, MICH.

(New York Meeting, February, 1916)

ALL the mining machinery of the Penn Iron Mining Co. has been operated by electric power for several years and when another shovel for stockpile loading was required the advantages of an electric shovel were naturally considered. After considerable study, serious objections suggested themselves the use of a shovel operated directly by electrical apparatus by reason of the complicated control, severe service on the motors

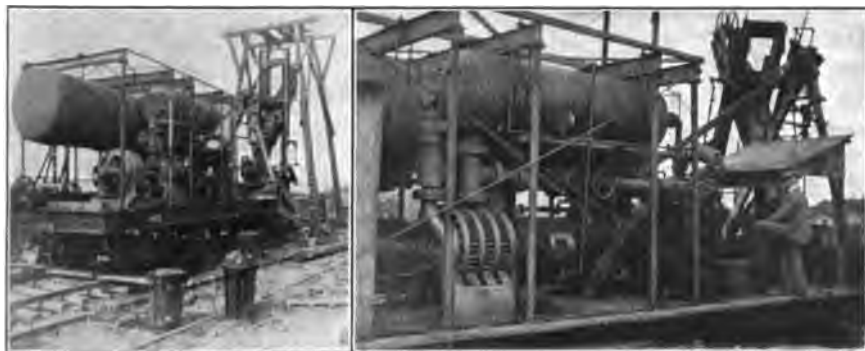


FIG. 1.—ELECTRO-HYDRAULIC SHOVEL
UNDER CONSTRUCTION.

FIG. 2.—THE SHOVEL AS IT WAS OPERATED
THE FIRST YEAR.

and the heavy surges of current required from the line. It was finally decided to construct an electro-hydraulic shovel using water under pressure to perform all the necessary operations except that of propelling, for which a separate motor was best suited.

A car body with boom, dipper handle and dipper of late design, but without any of the steam machinery, was purchased. A motor-driven centrifugal pump, a pressure tank, an air tank, a small air compressor and water cylinders with plungers, pistons and valves, were installed in place of the steam equipment. Fig. 1 shows this shovel under construction and Fig. 2 shows it as it was operated during the first year, the summer of 1914. The large tank at the back of the car, in Fig. 1, carried air under

* Mechanical Engineer, Penn Iron Mining Co.

a pressure of 275 lb. The smaller tank, seen best in Fig. 2, is the water-pressure tank into which the pump discharged and from the bottom of which the water was taken by the main header to the valves. Back of the water-pressure tank is another tank, the end of which can be barely seen in Fig. 2, used as a suction tank, which received the water exhausted from all the cylinders and from which the pump drew its supply.

The shipments of ore during the summer of 1914 were so small that the new shovel was not given a thorough test, but during the summer of 1915 the shovel requirements were more favorable. It was soon found that the tanks and the air compressor could be dispensed with, since a sufficient capacity could be obtained without the use of compressed air, by reason of the fact that the capacity of a centrifugal pump increases rapidly as the head decreases. The removal of the tanks improved the



FIG. 3.—ELECTRO-HYDRAULIC SHOVEL WITH COMPRESSED-AIR TANKS REMOVED.

appearance of the shovel as well as lowered the center of gravity. Fig. 3 shows the shovel after the changes had been made.

The dipper is hoisted by means of a large cylinder and plunger. The double hoisting ropes pass around two sheaves at the outer end of the plunger. One end of these ropes is anchored to the front flange of the hoisting cylinder, while the other end is fastened to the dipper. The dipper travels 2 ft. for every foot the plunger travels. The cylinder is single acting, since the weight of the dipper pulls back the plunger when the valve is open to exhaust.

The swinging of the boom is effected by means of a double-acting cylinder with a piston and rod extending through each cylinder head, with a sheave at each end of the rod. The swing circle at the base of the boom is moved by a rope, the middle of which passes around the front

end of the swing circle and the ends go around the sheaves on the ends of the rod and fasten to the car body.

As shown in Figs. 3 and 5, the thrusting is done by the piston rods of four thrusting cylinders directly connected to the dipper handle. The four cylinders give a perfect balance around the shipper shaft. The piping to the thrusting cylinder is made with swivels and sleeve joints, so that the dipper can be raised or lowered, or the boom swung either way or raised without putting any strain on the piping.

The shovel is operated entirely by one man as shown in Fig. 4. His right hand does all that the craneman on a steam shovel does. He pushes



FIG. 4.—ALL SHOVEL OPERATIONS CONTROLLED BY ONE MAN.

the lever from him to thrust out and pulls it toward him to bring in the dipper. His hand if dropped a trifle strikes the button shown below the hand and trips the dipper door. His left hand operates the valve for the hoist; forward to hoist, toward him to lower. His feet operate the swing valve by pushing on levers that are interconnected. By pushing on his left foot he swings the boom and dipper to the left, by pushing on his right foot he swings them to the right. When the feet are even, the valve is in the stop position to which the valve is brought automatically by a centering device. The handle of the controller for the motor which propels the shovel is seen under his right elbow.



FIG. 5.—DIPPER DOWN, WEIGHT CAUGHT.



FIG. 6.—DIPPER OVER CAR.

Considerable manual effort is avoided by the solenoid tripping device for the door of the dipper. This is shown in Figs. 5; 6, 7 and 8. A weight on a long arm suspends from a horizontal shaft and tends to hang vertically, being parallel to the dipper handle when the dipper is down. A small catch holds the shaft from turning as the dipper is hoisted, and when the dipper handle is horizontal the weight is ready to fall as soon as the catch is tripped. On the same horizontal shaft is a short arm the end of which is connected by a chain to the latch of the dipper door. When the weight falls, the jerk with the leverage which the weight exerts upon the short arm pulls the chain, and trips the dipper door. The small catch is released by a solenoid operated by the button under the operator's right hand. Fig. 5 shows the dipper down with the weight



FIG. 7.—WEIGHT RELEASED, DIPPER DOOR OPEN.

caught; Fig. 6 shows the dipper over the car with the weight ready to fall; Fig. 7 shows the weight tripped and the door open.

The average speed of the machine is between three and four dippers a minute. Repeatedly, 3,000 tons have been loaded in a 10-hour day, although the railroad service required shifting every two or three cars. Much more could have been loaded if good car service could have been obtained. With some slight changes it will be possible to increase even this speed considerably.

The power required and the speed of the shovel are shown in the chart, Fig. 9, from a recording wattmeter.

This shovel has few moving parts as compared to either a steam shovel or a straight electric shovel. There are no gears, clutches, brakes or drums. The few moving parts, except the motor and pump, move very

slowly. The pump gives no trouble since the water is clean and mixed with a lubricant. The leakage is small so that a hydraulic oil that will

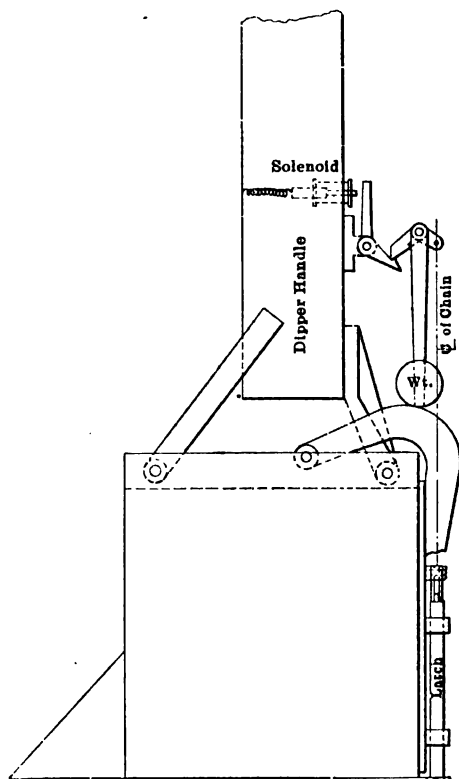


FIG. 8.—DIPPER-DOOR TRIP.

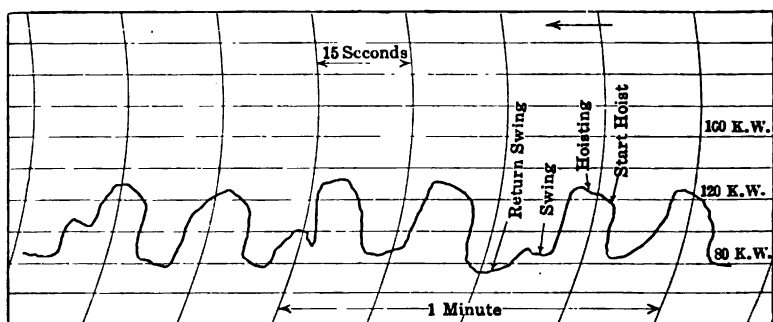


FIG. 9.—WATTMETER RECORD SHOWING POWER CONSUMPTION AND SPEED OF SHOVEL OPERATIONS.

not freeze can be used in winter. The only noise is the hum of the motor and a slight singing of the water as it rushes through the pipes and valves.

The advantages are that it is a one-man shovel not requiring a crane-man or a fireman. The current pull from the line has no large peaks and when the shovel is not in operation the current can be shut off entirely and the pump stopped. There is very little to wear out except the dipper lip and the packing. The control of the movements of the shovel is smooth and accurate. If it is desired to raise or lower the boom for rail transportation it can be done in a few minutes by resting the dipper on the ground and either thrusting or the reverse. It is practically noiseless and clean.

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Conservation and Economic Theory*

BY RICHARD T. ELY,† Ph.D. LL. D., MADISON, WIS.

(New York Meeting, February, 1916)

I. CONSERVATION DEFINED AND DESCRIBED

Conservation Means Preservation, Improvement, Justice

CONSERVATION, narrowly and strictly considered, means the preservation in unimpaired efficiency of the resources of the earth; or in a condition so nearly unimpaired as the nature of the case, or wise exhaustion, admits. And broadly considered, it means more than the word itself implies, for it naturally includes an examination of methods whereby the natural inheritance of the human race may be improved; and still more broadly considered—and as used in popular discussion—it includes a treatment of the effects of productive conservation measures upon distribution. We shall give our attention briefly to the main points in this informal definition before we pass on to the thesis that conservation is largely a matter of property relations—that a wise conservation policy means wise property relations.

Preservation of Natural Resources

First, conservation suggests simply preservation, and, in the treatment of the subject which we have had in the United States, emphasis has been laid upon the past and present waste of natural resources and upon the means of putting a stop to this waste. The forests have been depleted; soil has been washed by rapidly flowing surface water from mountain sides. These rapidly flowing waters have produced high streams and devastating inundations, followed by low streams and impeded navigation. The forests have, moreover, been so removed as to bring great danger of fire, and destructive conflagrations have resulted in large waste of resources, including in some cases a destruction of the humus, supplying essential elements in the fertile soil. As a remedy we must have measures to reforest mountain sides, and this means large social control, very frequently taking the form of public ownership and management; but in addition we always find actual and proposed

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† Professor of Political Economy, University of Wisconsin.

measures of coöperation with private owners. Illustrations are afforded by the services of publicly employed foresters in giving advice and in making plans, etc. Here the main emphasis is on cessation of waste and restoration of natural advantages. This means a great deal and involves hard tasks. It is said that there are places in southern Europe where it will take 300 years to restore the soil washed from the mountain sides, as it is necessary to begin at the bottom and by planting trees gradually to build up a fertile soil from falling leaves, decaying vegetation, etc.

Another feature of the existing situation much discussed is the enormous waste of good soil by the erosion of water, even in the more level portions of the earth's surface, as in the Mississippi Valley, where millions upon millions of tons of previous elements of fertility are being annually wasted—washed into this turbulent stream. It has been proposed to construct at public expense mighty reservoirs to check the rapidity of flow and to hold back the soil, which would also facilitate navigation. Here is one suggestion, and without entering into the extent of its possibilities, it is again apparent that emphasis is laid on preservation as nearly unimpaired as possible, as careful an exhaustion of supply, as the nature of the case admits.

Improvement of Natural Inheritance

But, in the second place, we are concerned with measures to improve the natural inheritance of the race, and this is brought out more or less fully in discussions of conservation. Under a rational system of forestry it is claimed that an acre of ground will produce two or three times as much in a given period of time as the same acre in a wild state of nature, when the trees are allowed to grow up naturally without man's care. Conservation of water adds to the fertility of the soil or at any rate brings about a better utilization of the fertile elements when the water is applied to the land in irrigation. Modern agriculture, which is at least partially included in discussions of conservation, is not content with the maintenance of an existing status, but wants to improve what is. Nearly everything needed to accomplish this purpose may be found in the land itself, or placed there by cultivating green crops and plowing them under, and perhaps extracting from the air elements of fertility to be united with the soil. The phosphates and potassium seem to be the main, if not the only, elements which must be added from without and which are in no physical proximity to the land, generally speaking. It must be admitted that the supplies of these are far more limited than we could wish, although it is highly probable that now unknown sources of supply will be discovered. Certainly there is no ground for present alarm, and for this assurance there are several reasons. With a general

appreciation of the natural scarcity of these elements, economy in their use will be practised to a greater extent than previously, and this economy will be further promoted by an increase in price, accompanying growing scarcity. The natural result will be to establish something approximating a cycle, whereby these elements can be returned to the land again and again. We see something of this sort in countries like China and Japan. We are, therefore, warranted in the belief that for an indefinite time to come, certainly for several centuries, the productivity of the land may be maintained and even increased.

Conservation Concerned Primarily with Production

In the third place, we observe that conservation considered as a part of economics has to do first of all with that division of economics which we call production; not with production in the technical sense, but in the social sense; in other words with relationships arising out of the efforts of men, associated to produce wealth. The question here has to be asked what institutions or what shaping of institutions is most favorable to conservation? Here, as elsewhere in economics, we have to do with property and contract, and find we cannot make much progress until we have adopted the social theory of property and the social theory of contract.¹

Justice in Distribution

And, in the fourth place, conservation takes cognizance of distribution and aims to bring about justice, as the conservationists see it. In general, it may be said that the conservationists wish to cut off, or at least reduce, the private receipt of property and income beyond what is a fair return to capital and labor and enterprise, reserving the surplus for public use.

Notice that we now get beyond the limits of conservation, strictly speaking, and are quite within the field of economics, but it would be arbitrary to stop the discussion at this point, where it yields fruitful results. This simply shows that conservation is in large part economics, one of our fundamental theses.

Conservationists seek to control franchises for the use of waterpower with this end in view, exacting payments for the public treasury in cases where otherwise we would have surplus value. They wish also a complete public control of mineral treasures, brought about by leases for the exploitation of these treasures when in public ownership, so that the lessees may receive no more than fair competitive gains; but some conservationists are inclined to look favorably upon at least a limited public

¹ The limits of space and time prevent an elaboration of these theories, and this is the less necessary as I have discussed them elsewhere. See my *Property and Contract*, Book I, Part I, Chap. VI and Part II, Chap. V.

operation of mines as well as ownership. The distribution of wealth and income is then always included as an aim in thorough conservation discussions.

Conservation Includes Two Orders of Inquiry. The First Belongs to the Natural Sciences

But when we reflect seriously on the subject of conservation, we see that we have to do here with two orders of inquiry: One of them falls within the broad field of the natural sciences; the other is economic in nature and is concerned with property relations.² The geologist must instruct us about the formations of mineral treasures, about their amounts so far as actually known and so far as they may be estimated, about exhaustibility, etc. The scientific agriculturist deals with the cultivation of the soil and discusses methods for the maintenance and even increase of agricultural productivity.³ Men who understand the natural laws governing the growth of trees are absolutely required if we would frame a wise forest policy. It is a question of natural science to determine the relative amounts of forest products which can be expected under low and middle and high forest culture, all involving different periods of time.

The Second Order of Inquiry is Economic

But chief emphasis must be laid on the too much neglected question of property relations. Is the system of leasing public land to private parties favorable in the case of mines? We have to do in this case with the relations of public property to private operation and to private property in the implements of operations. Shall the federal government, the States, cities and other local political units acquire private lands for forestry? Here we are concerned with the extension of public property in land and the corresponding contraction of private property in land. How shall we encourage oyster culture? Shall we make the beds of bays and rivers and bodies of water where oyster farming is feasible private property? If so, we extend private property and contract correspondingly public property. Shall we reserve public property in underwater land and encourage oyster farming and similar production of other sea-food by a system of leases which will encourage private investments of capital? Again, we are concerned with property relations. It is in the property relations most suitable for conservation

² It is recognized that in certain of its features, conservation has to do with political science and sociology. Here and now we must confine ourselves to economics as that one of the social sciences with which we are chiefly concerned in conservation.

³ The maintenance of productivity and conservation of elements are two different things, but it would take us too far into scientific agriculture to discuss this distinction here.

that the chief difficulty arises; and it is on this account that the chief rôle in conservation belongs to the political economists, who must cultivate more diligently than heretofore that part of their field which we may designate as economic jurisprudence. When we decide upon what we want so far as property relations are concerned, engineers and other technical men will be able to carry out the desired policy, and political science, including administration, must be called on for aid. Oyster farming will flourish when we devise and put in force right property relations, and it will furthermore flourish without the establishment of injurious private monopolies.

Irrigation furnishes also an excellent illustration. The government of the United States has encouraged technical investigation. Much effort, and very properly, has been put upon ascertaining the underground flow of water and what is called the duty of water, *i.e.*, the work which a given quantity of water may accomplish, the aim being to avoid waste and accomplish the maximum of effect with the minimum of effort. The questions of when and how to apply the water are involved here; and many other technical questions, some of them purely engineering, some of them primarily agricultural, could be mentioned. These are technical and essential. But litigation has been prominent and in some parts of the country has ruined many a farmer, resulted in mutual hatred of neighbors, and occasionally even bloodshed has been a consequence, while an appreciable proportion of the benefits of improved farming by means of irrigation has been swallowed up in the needless costs, direct and indirect. And the vital question of property relations was neglected in the meantime, one great department, the Department of Agriculture, at one time even taking a discouraging attitude toward economic study. However, this has all been changed under the efforts of the present Secretary of the Department of Agriculture. Now, when property relations are settled, the rest becomes comparatively easy.

The United States Government in the Reclamation Service has not neglected property relations so far as its projects are concerned, for they are based on a rearrangement of property relations in the interest of the public and the parties immediately interested. I refer here to the giving up of old "priorities" so as to promote a better distribution of water.

Did time permit, it would be instructive to follow this discussion of the nature of conservation with a brief historical survey, showing first of all what political economists have done for conservation, and how the ground was prepared for the modern conservation movement; secondly, describing the connection which has existed between the work of scientific foresters, such as Dr. B. E. Fernow, Gifford Pinchot and others with the conservation movement; and thirdly, outlining the services of those men, who under the leadership of Ex-President Roosevelt

made the program of conservation a vital force in the United States. But as it is, such a historical sketch must be omitted; and so we proceed at once to a discussion of some leading economic principles of conservation.

II. SOME ECONOMIC PRINCIPLES OF CONSERVATION

Conservation in its economic aspects has such a wide sweep that it is difficult to lay down many principles of universal application, but several can be elaborated which have very far-reaching applications. But before we call attention to certain general principles, we ask on the very threshold of our inquiries that you should consider this simple question, "What is waste?" The question is simple; the answer is in many cases extremely difficult. The Century Dictionary has over two columns devoted to the word waste as noun, adjective and verb, but does not satisfactorily answer our question. Under the verb, I find this: "to expend without adequate return." That is, at any rate, helpful. You see apples decaying on the ground in my orchard. You ask, "why this waste?" If I reply, "It will take two dollars worth of time to gather one dollar's worth of apples," are you satisfied that there is no economic waste? But this is precisely the case with a great deal of what the conservationist frequently calls waste. Now I shall not attempt a definition of economic waste, but I hope that the remainder of this paper will throw some light on the question, and will show that often we must choose between destruction of a proportion, larger or smaller, of natural resources and a loss of human well-being; and as the natural world exists for the human world, we know how we must decide when the issue is clear-cut between the two.

Conservation means in a large number of cases what economists usually designate as intensive as opposed to extensive use. Now intensive use means in general high price. This is true in agriculture—excellent fences, careful cultivation of the corners of field and the edges near the fences, if there are fences; also fields entirely free from weeds, which means the hoe as well as the horse cultivator. It means going beyond the point of increasing returns to decreasing returns in most cases. Nearly if not quite everywhere in the United States the intelligent farmer stops when to go further decreases his net money income; this means in a new country very often that the hoe is never used, that the edges of fields are ragged, that fences are not so neat in appearances as esthetic interest would demand.

*The Higher the Price of Land, the Better the Farming in the Absolute Sense.*⁴ Warren in his *Farm Management* comes back again and again

⁴ It is the high price of land rather than the high price of the products from the land that is the cause of good farming and intensive farming. It is important to bear this distinction in mind, but we cannot here enter into all the refinements of theory which flow from this distinction. Historically and generally, it is believed that the two go together.

to high price as a cause of good farming. A soiling system, *i.e.*, bringing the feed to cows, is conservation in the sense that it means more return per acre, but Warren says truly, "The labor of hauling the feed and manure, to say nothing of the cost of growing the crops, would more than pay the pasture bill on most dairy farms. It is evident that land and milk must be very high in price, before a soiling system will pay" (p. 179). And we find the following observation in the chapter on "Maintaining the Fertility of the Land:"

"We have good years and poor years, but crop yields are increasing very rapidly. All that is necessary to have them go up still farther is to pay the farmer more for his produce. By bringing in land that is now little used, and by better methods of farming, that are already known to farmers, it would probably be possible to increase our total production of crops 50 per cent. in three years if the farmer could be assured of prices high enough to warrant the expense involved."⁶

Undoubtedly it is true that without an increase in price, something may be done to increase yield. Sometimes farmers are more intensive than they can afford to be, but generally not so much so. The following from Warren is a correct generalization. "The farmer's problem is to intensify his business up to the point of greatest profit for his conditions. Since conditions are gradually changing in favor of more intensive methods; and since there is a tendency for the average person⁶ to lag behind, it follows that a little more intensive methods than the average of the community will usually be best" (pp. 179-180). But observe methods are changing to favor intensive farming because land prices are higher.

Another thing to be noticed is that improvement in knowledge and art increases the profitable intensivity of farming. We learn how to increase yields at a given price.

In what has been said we have a partial explanation of the intensive dairy farming which may be seen in parts of Europe, *e.g.*, near Frankfort-on-the-Main. But we have only a part of the explanation. Low wages are another part of the explanation; for, other things being equal, the higher wages are, the less intensive is that farming which is profitable; the lower wages are in a given state of the arts, the better the farming which is profitable.⁷ High wages encourage the use of machinery, and this is a partial offset, but only a partial offset to high wages. If wages could be cut one-third and the labor supply increased correspondingly, we should witness an improvement in farming which would be in the direction of conservation of natural resources and would please the

⁶ Warren: *Farm Management*, p. 183-184.

⁶ It is not a question of averages. Many lag behind and for all these better farming would be profitable.

⁷ High prices and low wages result in increased intensivity by drawing rent up. It is high rent with high land values that makes intensive farming profitable.

esthetic eye; but observe it is a conservation of natural resources which is at the expense of human resources.⁸

The Conservation of Human Resources Limits the Conservation of Natural Resources

Now this can be illustrated endlessly, in mining, as well as in agriculture. If we save all the ore taken from the mines in the Mesaba region in Minnesota, the price of iron ore must increase very materially or wages must fall, or both effects must be produced and either one means lessening real incomes of the community at large. As in farming, somewhat more economical methods could doubtless be employed at present prices for ore and present wages; but at best great waste of natural resources must continue, i.e., a great deal of the ore must remain unutilized.

Conservation of natural resources everywhere finds its sharp limitations in human resources, i.e., in human well-being.

Once again, conservation means a sacrifice of the present generation to future generations, whenever it is carried far, this conflict beginning far before the point is reached which conservationists are inclined to advocate.

This is implied in what we have already said. High prices and low wages, or even high prices with present or higher wages, mean a sacrifice of the interests of the present consumer. How shall we balance the interests of the present and the future? If we put them on absolutely the same plane, we could with propriety forego all use of the exhaustible natural resources like petroleum and natural gas; it would be at any given moment of time indifferent whether or not we used them, and they might remain forever unused—a *reductio ad absurdum*.

Perhaps the best article on the economic theories involved in conservation is that written by Professor L. C. Gray, which appeared in the *Quarterly Journal of Economics* for May, 1913, under the title, Economic Possibilities of Conservation. Professor Gray finds "the real heart of the conservation problem" in "the conflict between the present and future" (p. 499). He says that "the primary problem of conservation expressed in economic language, is the determination of the proper rate of discount on the future with respect to the utilization of our natural resources." He then continues as follows:

"Some discount of the future, there must be. If society reduced the discount on the future to zero, the period of utilization would be increased to infinity; and there-

⁸ Economic theorists will want to elaborate further and qualify the statement. High prices and low wages would in some cases result in larger scale and less intensive culture; but what is said here holds as a general statement. Rents would usually increase. Movements of population must be considered if we are to make our theory complete; but to do this would mean an immense enlargement of the present paper.

fore, the amount of present use would become infinitesimal. Conservation as a single principle of action involves the equal importance of future wants and present wants. It requires that the want of the infinitely distant future shall be as important as the want of the immediate present. Conservation as a single principle of action is reduced to an absurdity" (p. 515).

This is precisely the conclusion we have reached. Any positive, statistical and mathematical solution is an impossibility, because the solution, as Professor Gray also points out, depends in the final analysis upon questions of individual social philosophy. What is the purpose of existence? the source and nature of ethical obligation? our duties to posterity? all questions of the gravest import and beyond the range of economic science. But taking as a basis the ethical notions and sentiments of normal men, or perhaps those somewhat above the average man, we can find guiding principles and helpful suggestions in economic theory which we must apply as best we can in concrete cases in their infinite complexity.

Now, as one of the first steps in conservation policies, we must classify natural resources, because they differ markedly with respect to the intensity of the conflict between present and future interests. Some natural resources may be maintained forever or at least indefinitely with use; others may possibly be increased indefinitely while being used; others show increasing scarcity and exhaustibility; and where supply is so sharply limited in proportion to demand that we begin to feel the effects of coming exhaustion and where no renewal is possible, we have the sharpest conflict between present and future. Let us for our present purposes adopt the classification of Professor Gray, which as made by an economist is especially adapted to the purposes of economic discussion.

"Natural resources may be classified as follows:

"I. Resources which exist in such abundance that there is no apparent necessity for economy, either in present or future. For instance, water in some localities.

"II. Resources which will probably become scarce in the remote future, although so abundant as to have no market value in the present. For instance, building stone and sand in some localities.

"III. Resources which have a present scarcity:

1. Not exhaustible through normal use: Water powers.

2. Necessarily exhausted through use, and nonrestorable after exhaustion: Mineral deposits.

3. Necessarily exhausted through use, but restorable: Forests, fish.

4. Exhaustible in a given locality but restorable through the employment of other resources of a different kind or of similar resources in different locations: Agricultural land"⁹ (pp. 499-500).

⁹ In a footnote Professor Gray says: "In terminology the above classification resembles one proposed some years ago by Professor B. E. Fernow: See his *Economics of Forestry* (p. 10). In detail, however, the classification differs widely" (p. 500).

Interest represents the difference between present and future to the individual. It has been said in the economy of the State, as in publicly owned forests, interest does not enter, that it is a concept in distribution of income among private persons; but, however this may be, we cannot avoid the recognition of differences between present and future. Can it be asked that we should do more than discount the future at the lowest possible rate of interest paid by a prosperous state with well-managed finances, say 2 per cent.? I throw this out as a suggestion. It is only a little bit of a sidelight on the complicated problem.

It is essential at this point that we consider the case of copper and other natural resources which are probably fundamental conditions of permanent national existence. Even if there is some doubt about this, so long as it is at all probable that any national resource is essential to the continued and permanent existence of the nation, this resource occupies a peculiar condition. It is a first principle of political science that the State has perpetual life. States have perished in the past, but political and economic science cannot take into account the possibility that our own national life will ever cease to exist. All wise plans must be based upon the hypothesis of continued national existence. Now in a case of this kind the future value of the natural resources rises to infinity, and however much we discount the future, we must still practise conservation. We must attempt so to utilize every natural resource of this kind that it may last as long as possible, using it now and using it hereafter for as long a time as possible. I think there in reality the interest rate cuts no figure. When we consider resources of this sort, we find that in their case the definition given by President Van Hise at the close of his book is applicable: Conservation means "the greatest good to the greatest number—and that for the longest time."¹⁰

There has been some discussion as to whether in agriculture we want the largest product per man or per acre. In general, it is doubtless true that the chief determinant should be the quantity produced per man. But the principle just laid down leads to certain qualifications. We have to consider not only the quantity produced per man now, but the welfare of future generations, and especially must we consider the condition of the country at war. The conservation of natural resources has to be regarded from a national point of view. No nation can consider it otherwise at the present time. From the German point of view we can see that it was important that Germany should sacrifice, as she did, something from the largest possible production per man to increase the production per acre. Conditions in Ireland also show that in the interest of present and future generations we cannot decide solely with reference to the largest production per man.

¹⁰ C. R. Van Hise: *The Conservation of Natural Resources in the United States*, p. 329.

Adequate conservation in general means a course of conduct in economic affairs which is dictated by the common interest, but which will not be followed by the private persons under a system of *laissez faire*, or nonintervention.

The complaint of the conservationists is precisely this: that the individual does not, as a matter of fact, conserve natural resources. What is going to induce the private person to follow the desired course of economic conduct?

We consider among various possibilities this: the individual through ignorance fails to follow a socially desirable course of economic conduct, when it would be in his own interest. We have already seen that often, perhaps in most cases, the individual could be more careful, more painstaking, more intensive in the sense of conservation—for not all intensive uses mean conservation. We have, as a remedy against ignorance, education, and education can discover many new ways of conserving, to a greater degree than at present, all natural resources.

Even with respect to the present, education, if properly directed, will result in a diminished waste of economic goods. Psychical development fails to keep pace with economic changes. A highly educated woman, a missionary who had long worked in Turkey-in-Asia, was asked by a ladies' society in Buffalo what impressed her most on her return to her native land, and she replied, "Your garbage pails." She saw in them sinful waste—good food thrown away in a single city, which would nourish thousands and perhaps even ten thousands of half-famished people in Armenia. Several causes of American waste in this particular can be mentioned. One is that when food was superabundant, that was not waste which is now wicked waste. It is not wasteful to feed potatoes to pigs when the supply is greater than the needs of all human beings who can be reached. We have in general left behind us the days of crude plenty, but have not changed our psychical make-up nor our habits to correspond with new economic conditions. Here the need is intellectual and moral education—a better vision and more altruism. We need a keener social consciousness and a new state-sense, if we are ever to solve the problems of conservation.

The individual is too frequently thoughtless and indifferent with respect to the future. Education of the intellect helps to a slight extent, because as men become more intellectual, the future means more to them. Moral education helps still more, as it means precisely a strengthening of the other-regarding feelings as opposed to self-regarding impulses. It means more self-control and ability to sacrifice within rational limits the present for the future. Better government means that the future counts for more, as it is more certain. Better health and long life further decrease the premium which the future must pay for present sacrifices. Without adding further considerations we may generalize

as follows: Every step forward in civilization means increased regard for the interests of the future.

But, on the other hand, we must acknowledge that enlightened self-interest has a greater rôle to play in conservation than is generally understood. When attention 15 years ago was called to the enormous waste of natural resources, the private owners began to think about the possibilities of profitable conservation, and they are still making investigations and improving methods. Great mining companies particularly may be mentioned as illustration, and naturally the owners of the fee in mining properties have shown a special concern. Real progress has thus been made; but the rate of interest sets a limit, as my colleague, Professor Leith, has shown. Some ore companies have been obliged to recede in part (in part only, observe) from earlier conservation practices, because they found no prospect of getting interest on the investment involved in carrying saving so far as they were doing.

Large concerns are apt to be operated by far-seeing men, who acquire at times an interest of affection, so to speak, in these concerns, and they will be inclined to go as far as is practicable in conservation; as the country grows older, the interest rate is likely to fall and other causes will favor an increasing degree of conservation, due to private initiative.

But there is a sharp limit to the economic sacrifice that we may reasonably and ethically ask the private person to make for even the present welfare, and the limit is still sharper when we come to consider the interests of future generations. When it is possible and as a general principle, social burdens should be socially diffused and socially borne.

Let us consider some application of the principle we have just formulated. We know now that it is one of the prime functions of the State (State used in the generic sense and including federal government as well as the separate commonwealth) to raise the ethical level of competition by legislation and administration. Men may work 12 hours a day under the system of *laissez faire* and their employers may compete with each other. If, by the State, the number of hours is reduced to eight, the competition of employers may continue unchanged, save that it takes place on a higher ethical plane. Endless illustrations could be given, taken from Sunday work, child labor, sanitary conditions, etc. Now we may similarly establish a level of competition based on the principles of conservation. Practices which are not in accord with that level of conservation, which we wish to establish, may be prohibited. When we decide how carefully the coal should be mined, and how much waste of natural resources in the exploitation of iron ore we are willing to tolerate as a maximum, we may prescribe methods giving the desired results, and we must bear the pain or burden in higher prices. Competition

would not allow one individual to increase his costs greatly above those of his fellows; and so common action is a necessity.

The mention of competition suggests that unregulated and uncontrolled it is very generally speaking destructive alike of human and natural resources, and that it is a force which like the flow of rivers needs to be bridled and guided to produce socially beneficent results. Destructive competition can be prevented only by social efforts. Again, I must mention the work of President Van Hise in pointing out in his book, *Concentration and Control, A Solution of Trust Problems in the United States* (1912, revised edition 1914), the evils resulting from those "trust-busting" campaigns of politicians who advocate competition at all times and places and let loose upon us forces of destruction.

But as the principle of regularities in large numbers forms the basis of the vast business of insurance, it also furnishes a considerable scope for the activity of the State in the conservation of human natural resources. It is well known that individuals, especially among the wage-earners, are too willing to take risks to life and limb. Let us suppose the chance of becoming a helpless cripple in the course of a year is as 1 to 10,000; this seems so small that the individual is generally ready to overlook it, and neglects measures to remove the danger. But to society there is no chance, only certainty of this human loss, which in its ramifications is far greater than is at first apparent. It means 100 wrecked existences per million and 10,000 per annum in a population of one hundred millions. We have in this principle of regularities in large numbers a wide scope for society to promote conservation alike in its own interest and in the interest of the individual. The "Safety First" movement is one of many evidences of an awakening realization of possibilities in this direction.

As another illustration of the possibilities of altogether wholesome conservation which comes under this general head I mention the enormous destruction caused by fires in the United States. Public authority must compel those precautions which are alike in the interest of the individual and of society.

But we may look at this question from still another angle. *Protectionism has often an educational value.* When we prohibit certain practices which exhaust natural resources rapidly, it will often be found that through the pressure of legal force we shall be stirred out of a certain inertia of lethargy, and discover better methods which involve little or no increase in expense.

But we find ourselves in a position when we shall frequently have to decide between private property and public property. The benefits of private property come in large part from the free initiative it allows the private property owner, and this is based on the hypothesis that in following his own initiative and knowing his own interest he is following

a line of conduct which is in the interest of society. This is illustrated by the condition of the farmer, where we have a fairly satisfactory system of land tenure. The intelligent farmer in pursuing his own interest, in raising the best crops and animals at a minimum cost, is doing exactly what it is in the interest of society he should do. The case of our railways is the opposite, for we regulate these to such an extent as to remove from private property a large part of its satisfactions and its benefits. So if we are obliged to regulate very far private property in the interest of conservation, we have a strong ground for public property; as illustrated in the case of forests, and in this case public ownership is the world over gradually gaining on private ownership. Consideration of conservation then leads us to the following:

"The principle of guidance in changes from private to public property and from public to private property.—Private property yields the best results when the social benefits of private property accrue:

(a) Largely spontaneously.

(b) When occasionally they are easily secured by slight applications of force.

(c) When the social benefits of private property are secured as the results of single public acts occurring at considerable intervals.

(d) Private property may yield excellent results, when in more or less frequent cases a continuous and considerable application of force may be needed to bring its management up to a socially established ethical level.

"But in proportion as the social benefits desired are secured by increasingly intensive and increasingly frequent applications of public power, the advantages of private property become smaller as contrasted with the advantages of public property."¹¹

We have furthermore a ground for passing over to public property when for the sake of the public welfare now or in the future a kind of use is exacted, involving a greater sacrifice of individual and corporate private interests than is warranted by the doctrine of the police power, as accepted at a particular place and time.

I cannot go into all these points, but I am confident that the more you think about this statement of a general principle, the more far-reaching you will see that it is in its application to conservation. I want to say, however, just a word about the last clause, which relates to the police power. We hold all our private property subject to that; in other words, certain general sacrifices may be exacted of all in the general interest, relating to health, decency, etc., and this burden increases as time goes on; but when we must exact more than is in harmony with this limitation, known as the police power, we have a ground for public ownership.

¹¹ See Ely's *Property and Contract*, vol. i, p. 352, where an elucidation of this principle is given.

No one is warranted in jumping to the conclusion that this principle means a suppression of private ownership of land and capital and of private initiative. On the one hand, our classification of natural resources shows that, so far as we can now see, they are mostly adapted to private ownership; we have also observed improved practices of private owners, rendering less urgent and less far-reaching the demands of public ownership; on the other hand, the growth of the police power in itself makes the field of public ownership narrower than it would be otherwise. Yet the general principle holds and no one can now foresee where the line will be drawn in the future. In the case of forests, the civilized world now recognizes a large amount of public ownership as necessary; so also with regard to the shores of harbors; so also with regard to mineral treasures, where there is general opposition to further alienation of the fee where it is now in public ownership.

Conservation then necessarily means more public ownership, more public business; this means a demand for better government; and this means giving men a real career in the public service.

It would require a small book—perhaps a large one—to elaborate the principles I have hastily sketched, and to fill in the gaps and complete the argument would certainly require a very big book. But I trust I have outlined the field for the economist in conservation.

Commissions Alone Equal to the Tasks of Conservation

Now I will close with one very practical suggestion. It is only through a conservation commission or various conservation commissions—for several may be required—that we can put in force conservation. Legislation can never solve these complex problems, but can simply lay down the general principles, expressive of the will of the legislature; and in concrete cases the commission must make the application of principles, in other words, ascertain the will of the legislature. The commission must ascertain what is excessive present use and what is waste, must in concrete cases weigh over against each other present and future, must decide upon the burden to be imposed on private property under the police power, and upon compensation which is feasible and required; it must set limits to the sacrifices which may be legally and ethically exacted of the individual and the private corporation.¹² From time to time it will have to recommend the establishment of new principles by legislation; and the courts, as having the last word in social progress, will review certain decisions.

¹² On the sense of social responsibility in its relation to conservation and the sacrifices to be enforced by law on the individual and the private corporation, see Van Hise: *The Conservation of Natural Resources in the United States*, pp. 362-3; 374-5; 377. I think President Van Hise exacts too much of the private individual.

But the great burden must rest with commissions. And do not commissions give us the democratic solution of the complex economic problems of our day? The legislature lays down the principles, and if the commissions furnish men with careers, with honor, the public may command as well-trained capacity and as high talent as the greatest private corporations and for far less cost. Recognition, the development of democratic sources of honor, open careers to capacity and talent all these will draw to the service of the people men who will be equal to the tasks of government, and who in their own persons will illustrate the nobility of social service and make men proud to say: "I am a civil servant."

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Conservation of Iron Ore*

BY C. K. LEITH, MADISON, WIS.

(New York Meeting, February, 1916)

QUOTING from Dr. Richard T. Ely:¹

"Conservation, narrowly and strictly considered, means the preservation in unimpaired efficiency of the resources of the earth; or in a condition so nearly unimpaired as the nature of the case, or wise exhaustion, admits. And broadly considered, it means more than the word itself implies, for it naturally includes an examination of methods whereby the natural inheritance of the human race may be improved; and still more broadly considered, and as used in popular discussion, it includes a treatment of the effects of productive conservation methods upon distribution.

"Conservation means a sacrifice of the present generation to the future generations whenever it is carried too far. There is a sharp limit to the economic sacrifice that we may reasonably ask the private person to make for even the present welfare, and the limit is still sharper when we come to consider the interests of future generations."

In a paper on the iron ores of America, presented to the Pan-American Scientific Congress at Washington, I have emphasized the fact that the reserves of iron ores in the two Americas are enormously in excess of the requirements of the present generation, in fact, so large that no shortage is to be anticipated for many generations in the future; that a large part of the value of iron ores is put into them by man's efforts to make them available. This fact is fundamental for the intelligent consideration of policies of conservation as applied to iron ore. As I interpret it, this fact makes it practically unnecessary that the present generation should make any especial sacrifice for future generations in the way of conservation of its iron-ore resources. Even if we were disposed to look forward to a period of 100 or 200 years, with a view to insuring the welfare of the people living at that time, it is doubtful whether any procedure we could now formulate would materially help them in the matter of iron ores, so large are the reserves available. In order that an appeal for conservation may be effective it must obviously favor the welfare of the present or immediately following generations; it must promise benefits to general and individual welfare during a period within the range of comprehension of the average man.

* This paper is copyrighted and must not be republished.

¹ Richard T. Ely, *Economic Aspects of Conservation*. Prepared for Pan-American Scientific Congress, Washington, 1915-1916.

Within this more restricted field there is ample opportunity for conservation measures which can be made of practical benefit to the present generation and indirectly help the future. Such measures include a variety of mining and business methods which will tend toward the minimizing of waste of ores and bring the maximum possible return on the investment; in fact, all measures which are taken by any far-seeing business man or corporation looking forward to the maximum return, not for the present year, but for a series of years. The business man measures the relation between the present and the future profit by the interest rate. With this as a basis, he takes such steps as will insure the maximum return to himself or immediate posterity. To make such expenditures as will amount, with compound interest, to an aggregate beyond the probable return in the future, means a sacrifice of the present to the future which does not appeal to him as a business transaction and for which there is no strong demand based on possible shortage for the future. If, for instance, in mining Mesabi iron ore it is found possible to stock certain of the lower grades at such a figure that when compounded at a normal interest rate the money will be returned at some future time, the operator is using a conservation method which is a benefit to himself and immediate posterity. To spend for this purpose an amount so large that the receipts for this stockpiled ore will never catch up with the compound interest on its preliminary investment means sacrifice to the future, which the large reserves of ore do not seem to require. If the miner spends \$2 worth of energy and materials to save \$1 worth of materials for posterity, he is making a sacrifice too great to be required of an individual. Up to certain limits, it is often cheaper to waste than to save, that is, cheaper when the human factors in the equation are considered as well as the amount of material saved. In such cases wasting may be real conservation when public welfare as a whole is considered. The problem comes down to a question of how much one can afford to spend in order to accomplish a result which will eventuate during a period of years in the future with due regard for the interest rate, which measures the difference between the present and the future.

Such kinds of conservation as above indicated are aimed at in the normal course of enlightened business. The good operator of mines is distinguished from the poor operator to some extent by his ability to think far enough ahead. In planning for the future he often is aided by the demands of the fee owner who is interested in getting the maximum possible returns from the property through the life of the mine. In fact there often develops here a conflict of interest which may even lead to measures of conservation detrimental to the best interests of the operator. With the concentration of control in large corporations has come a natural tendency to think in larger terms and to apply conservation methods which will bring the maximum results through

a period of years representing the probable life of the corporation. In this the interest rate is figured as a matter of course. The well-established large company is not so much at the mercy of immediate requirements as competing smaller interests, but has the means to conduct its operations on the basis of average returns through a long series of years. It does not find it necessary to make a considerable sacrifice of the future for the profit of the immediate years. In this respect, concentration of control has undoubtedly exerted a conserving influence as compared with unrestricted competition working on the principle of *laissez faire*. On the other hand, over-capitalization requiring immediate large returns may be inimical to conservation of resources.

The natural tendency of business to seek a standard of operation determined more or less by the best results of the most far-sighted people will, as a matter of course, involve improvement in conservation methods which will benefit the present generation. The present interest in questions of efficiency and efficiency systems is a step in the direction of conservation, which is leading to an analysis of the factors in the problem and the determination of how far one can wisely go in sacrificing the present for the future in order to get the maximum possible returns. The development of economic theory is leading to a clearer recognition of the factors involved and therefore to plans of operation involving less waste, when the operation as a whole is considered, through a period of years.

Notwithstanding these tendencies, the average of operation of iron mines is far from reaching the standard of efficiency set by some of the more far-sighted and larger companies. There are far too many operations in which the results of the present or next year figure so largely in plans at the expense of future profits. It may be said that such operators are the only losers, but it probably does not require an extended argument to indicate that the general welfare is promoted by the welfare of the maximum number of individuals. There is, therefore, abundant opportunity for the application of educational methods designed to raise the general standard. Innumerable technical societies are disseminating information tending to improve the standard. Government organizations, like the Bureau of Mines, Geological Survey, and various State organizations, tend toward this raising of standard, by making available to all the best results and methods of the few.

The problem of conservation is so complex that, as the writer sees it, it is practically impossible to lay down rules of conservation applicable to iron ore as a whole, to say nothing of all mineral products. There is too great a variety of local conditions. What might be intelligent conservation in one case would amount to confiscation or needless sacrifice in another. It might be possible, for instance, in a given district where it is shown that a certain grade of ore could be conserved for the

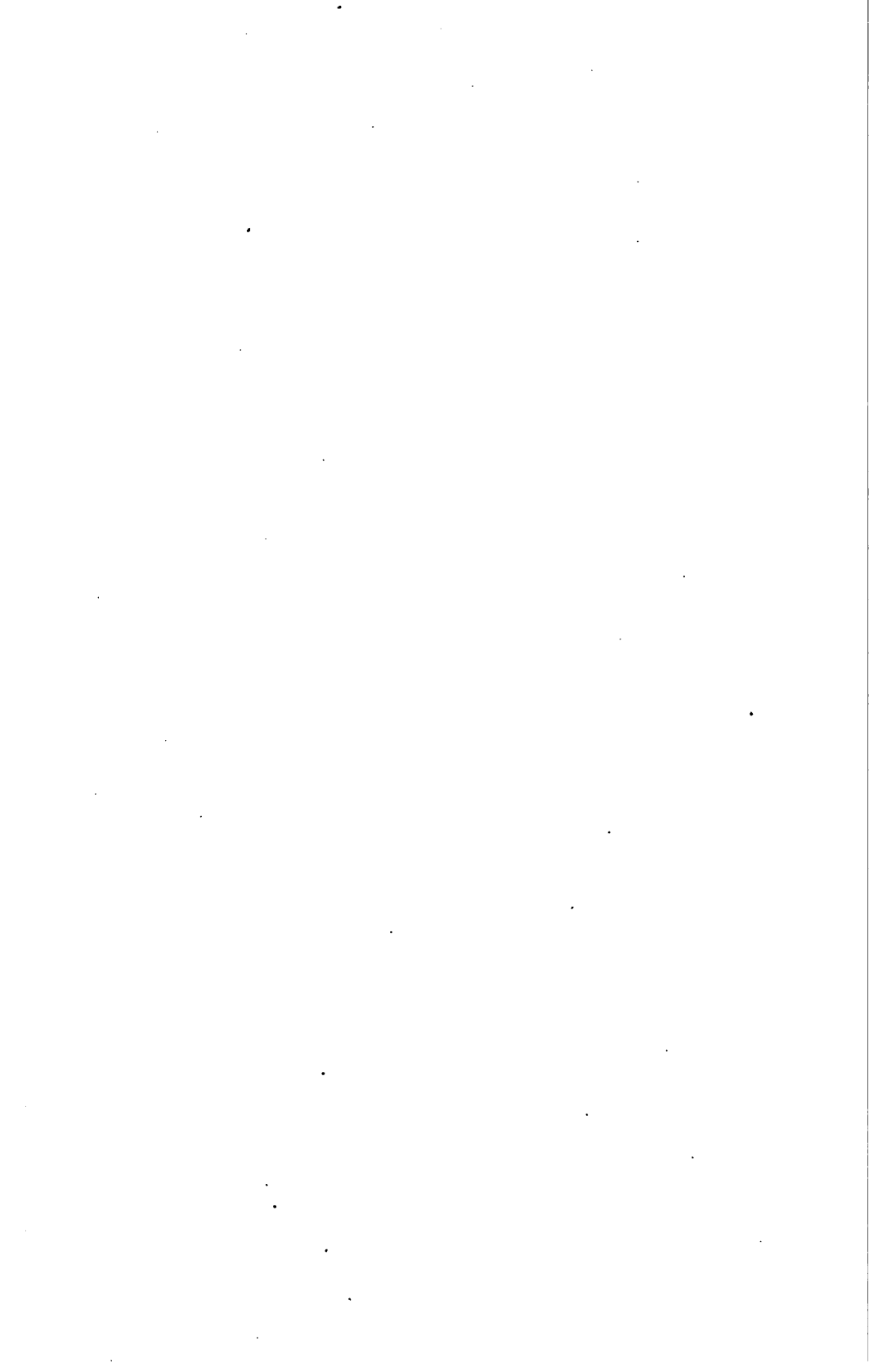
future for a certain expenditure, to formulate rules covering this. However, to carry this rule over to another region might require unwarranted sacrifice on the part of the individual in favor of future generations.

The conservation propaganda which has been so vigorously waged in recent years in the United States has, as a rule, not been qualified by the above restrictions. The emphasis has been entirely on the welfare of posterity, implying sacrifice to the present generation, without raising the question whether this sacrifice is in all cases warranted, and without attempt to balance the welfare of the present and the future. There seems to have been little recognition of the principle that measures tending toward the welfare of the present generation are conservation in their effect; are the natural first steps in any conservation plan; are the ones which can be most intelligently formulated; will indirectly benefit future generations, and will logically lead to the development of further measures for the protection of future generations when it becomes clear what balance between present and future welfare should be striven for. The result has been that many mining men, sympathetic with the general idea of conservation, have been repelled by its impracticability when applied to their specific problems. Others carried away with the enthusiasm of the idea have attempted to introduce measures which have become so burdensome and such a handicap in competition that they have been speedily abandoned. A number of cases might be cited of specific and distinct benefits of the conservation movement in initiating specific measures for conservation of ores. The most important result of the movement to date seems to have been to stir people to consider conservation questions and to develop a favorable atmosphere for the introduction of specific measures which on examination may be found to be practicable. It seems to the writer that definite recognition of the fact that conservation involves consideration of the welfare of the present as well as that of the future should be a powerful stimulus to development of effective conservation measures. The changes will be largely accomplished through an individual initiative in attempting to reach the highest business standard. Governmental and educational agencies can supplement effectively these tendencies by spreading the information and in making certain standards compulsory for specific groups of conditions.

For the government or other public bodies to go further, and to formulate rules for conservation which require sacrifice of the individual to distant generations, would necessitate a considerable exercise of public power, backed by strong public sentiment, which does not seem to be needed in the case of the iron ores, because of the enormous quantities available, which make it reasonably certain that posterity will be taken care of. The probability is that in the future, as in the past, the individual sacrifice will be limited to sacrifice of the immediate moment for

a gain which can be definitely approximated in the near future. If, however, it becomes desirable to bring public power to bear on the development of the iron ores, with a view to conserving these ores for posterity at a considerable additional expense on the part of present owners, the burden of such sacrifice should be taken by the government and not by the individuals. Quoting from Ely:¹ "When it is possible, and as a general principle, social burdens should be socially diffused and socially borne." Professor Ely² argues that at this point public ownership should be considered. "In proportion as the social benefits desired are secured by an increasingly frequent application of public power, the advantages of private property become smaller as contrasted with the advantages of public property."

¹ *Loc. cit.*



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A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron

BY W. McA. JOHNSON, HARTFORD, CONN.

(New York Meeting, February, 1916)

THE work of J. E. Johnson, Jr., on the effect of small amounts of oxygen in cast iron in increasing its strength and resistance to shock, is of interest from the technical and scientific standpoints. The following exposition of the theory carries Mr. Johnson's explanation further and, in my opinion, will disclose certain related phenomena and perhaps shed further light on the subject.

Some facts, so novel in character as to excite incredulity, have been accepted finally by many metallurgical authorities. One of these is that oxygen to the amount of 0.060 per cent. in cast iron gives it a breaking strength of 3,500 lb. per square inch, compared with a breaking strength of 2,500 lb. per square inch in cast iron of the same composition with respect to elements other than oxygen, but having only 0.010 per cent. oxygen. The "oxygenated" cast iron, even if made in a coke furnace, has properties equivalent to the best charcoal pig iron. Mr. Johnson has also established the fact that the variations in combined carbon cannot account for the results. By microphotographic evidence he has shown that in oxygenated cast iron the graphite particles are dense and spherical, while in well-deoxidized cast iron the graphite particles are flaky and leaf-like. It is just a matter of common sense that iron of the latter structure is much weaker than the former. The analytical and microscopical work has been done with care and there is no doubt in my mind about the accuracy of the experimental results. In this paper then, we shall consider it as accepted that small amounts (0.060 per cent.) of oxygen increase the strength of cast iron by producing a structure in which the round and solid particles of graphite are surrounded firmly by a principal matrix of ferrite.

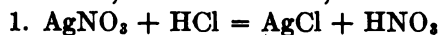
The purpose of this paper is to show that the reason why the graphite particles are round is founded on the fact that the particles of any precipitate are made denser and harder by the presence of a reagent having a solvent action on the precipitate. This can be stated as a law, although I have never seen it so given in any textbook.

Let us then, for purposes of illustration, give a concrete example and develop the theory of such mechanical and chemical actions as occur:

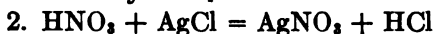
It is well known that, by a soluble sulphate, a barium salt in solution is precipitated in a dense and easily filtered body when the solution is strongly acid with hydrochloric acid, while in a neutral solution barium sulphate is thrown down as a flocculent body. The same is true of silver chloride precipitated out of a silver nitrate solution by hydrochloric acid with nitric acid as the excess reagent. If we increase the proportion of hydrochloric acid in the case of barium sulphate, and the nitric acid in the case of silver chloride, the precipitate is thereby made dense and easily filtered. In any case, the mechanism of the operation is the same, provided a sufficient interval of time be given to allow the mechanistic operation to proceed to equilibrium.

It may be remembered that this equilibrium is not a static, but rather a kinetic one. By this it is meant that an interchange of molecules from the solid to the solution is continuously taking place. We have the velocities of the two reactions that take place as equal and opposite.

We have, as is well known, the two equal and opposite reactions:



$$\text{Velocity} = V_1$$



$$\text{Velocity} = V_2.$$

To use an old-fashioned term, the influence of a solvent on a precipitate is as follows:

The first form of any precipitate is particles of molecular fineness. If these particles subsequently agglomerate, they agglomerate in long dendritic forms, in accordance with the probable effect of a weak force of cohesion acting on an infinite number of solid particles. In general, the velocity indicated above as V_1 is less than V_2 , and then the precipitate would be dissolved; on the contrary, whenever V_1 is greater than V_2 , precipitation would take place.

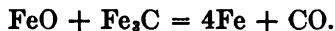
Considering now a case when V_1 is greater than V_2 and precipitation is taking place, and a particle having a body terminating in a point: It is obvious that the instantaneous effect of V_1 would be less on the point of such a particle than it would be about the center, on account of the law of mass action. Therefore, the particle would tend to be dissolved at the point and would then tend to be reprecipitated near the center of cohesive force of each particle. These interchanges would be comparatively rapid at first and then slower, but the general effect would be that the precipitate would become more and more composed of particles of such shape and density that the force of cohesion would act more strongly as the molecules came into closer mutual relation.

The net effect of this would be that the particles of spherical shape would increase in size and density according as the magnitudes V_1 and

V_2 were unequal instantaneously, or, explaining it in another way, according to the integrated heterogeneity of V_1 and V_2 .

According to the law of mass action, an excess of HNO_3 would increase V_2 as compared to V_1 , thus tending to convert the particles of irregular size into the form of a sphere.

Just this action occurs with oxygenated cast iron; we have the solvent action of oxygen on carbon with the production of carbon monoxide which is dissolved in the iron—we may even have the formation of iron carbonyl. We know that when liquid iron cools, graphite forms together with the eutectic. Any excess of oxygen above 0.010 per cent. would cause an ebullition of gas, because the solubility point of carbon monoxide in iron would be overreached and there would ensue the steel-making reaction:



It can easily be seen that the percentage of oxygen must be exactly that called for by the physical-chemical effect, and it is a great credit to Mr. Johnson that he has learned this by empirical methods. His deduction from *a priori* reasoning, based on premises calculated from many isolated phenomena, is a brilliant intellectual feat, as is his experimental proof.

It may be that further work on the effect of oxygen in cast iron will lead to the conclusion that a certain percentage of oxygen will improve steel. Possibly reagents other than oxygen might be used. As developed in this paper, the application of a chemical theory to the question of the size of particles of the several components of cast iron and steel has attractive possibilities in the domain of both pure and applied science. Unquestionably the presence of the right amount of a solvent will increase the density of a component, provided conditions are such that the law operates within proper limits.

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The Iron Mines of the Sierra Menera District of Spain

BY VICTOR DE YSASSI

EDITED BY A. S. CALLEN, SO. BETHLEHEM, PA.

(New York Meeting, February, 1916)

THE iron mines of Spain are located on the mountain ridge forming the boundary between the Teruel and Guadalajara provinces, called Sierra Menera. They form a property of 25 mines extending over an area of 1,677 hectares (4,140 acres) situated at a height varying from 1,200 m. (3,940 ft.) to 1,500 m. (4,920 ft.) above sea level. The deposit follows the direction north to south; huge masses of ore are being found both on the slopes and at the summit of the Sierra. The ore occurs among quartzites and magnesite limestone, in some places overlaid by quartzite 15 m. (50 ft.) thick; in some others (and this is more general) the ore is overlaid by earth and boulders of quartzite origin; this layer is about 8 m. (26 ft.) thick. The ore beds occupy the geological horizon of the Superior Trias or Muschelkask, which are here in direct contact with the Silurian quartzites. The lime-stones have been more or less transformed.

There is ample evidence to show that these mines were worked in very ancient times—objects and tools employed in mining by the ancient Romans and Arabs are to be found plentifully in old workings which extend down into the deposits. It has been estimated that more than 10,000,000 tons of ore were mined in the old days in this district and carried to the primitive forges in the vicinity where iron of much renown in Spain was extracted. Remains of more than 20 of these forges are found in the neighborhood of Sierra Menera and are usually situated near waterfalls and forests.

Much investigation has been carried on for the purpose of determining the quantity of ore to be found in these mines, but no exact calculations can be made at present. When the investigation is completed it will surely be found that the deposit contains a much larger quantity than the 100,000,000 tons indicated by the present workings.

The mines are now worked as open quarries and this method can and will be followed for a long time, probably until the deposit is worked out. Figs. 1 and 2 give some idea of the quarries in operation and

NOTE.—In this paper some data from *Fairplay*, Nov. 23, 1911, have been used in conjunction with material received from V. Ysassi.—A. S. Callen.

the reserves, the iron-ore stocks, screening installation and general arrangement.

From these quarries 1,000,000 tons can be mined annually (987,000 tons in 1913) and the production could easily be doubled if the market demanded it.

The ore known as *Sagunto Rubio* is carried directly from the mines to the boats without any treatment. The rest of the production is screened, giving two qualities—one of lump and rubble, the other of fines. The lump and rubble form the class called "screened Menara." This gives an ore of excellent physical condition. The fines, which are under 20 mm., are transformed into nodules or briquets.

The chemical composition of the different classes is shown in the following table:

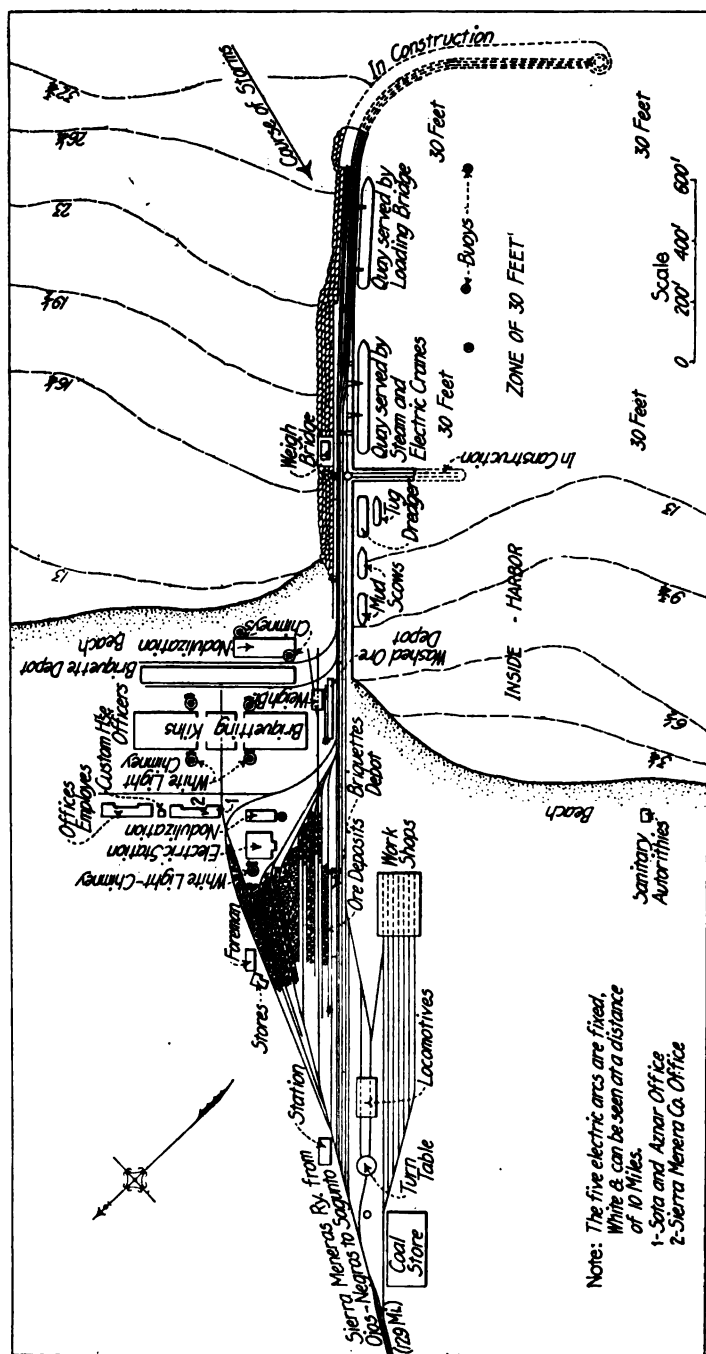
	Menara, Per Cent.	Sagunto Rubio, Per Cent.	Briquets, Per Cent.	Nodules, Per Cent.
Iron.....	54.300	53.500	62.000	62.000
Manganese.....	1.760	0.860	2.040	2.040
Phosphorus.....	0.025	0.105	0.030	0.030
Sulphur.....	0.013	0.014	0.015	0.015
Silica.....	5.160	9.840	6.010	6.010

Railway.—The railway over which the ore is shipped is 210 km. (131 miles) long, 1 m. gage, and extends from the mines to the port at Sagunto, running through the provinces of Teruel, Castellon and Valencia. The country traversed is mountainous and abrupt, which necessitated the building of important bridges and tunnels. The largest bridge is a great viaduct 48 m. (158 ft.) high.

The railway has a large capacity and is well equipped with locomotives and steel cars. The locomotives are of the compound type with six coupled axles with a weight of 14 tons on each. They were built by the North British Locomotive Co., Ltd. The cars are of the automatic hopper discharge type, with two axles, carrying 20 tons each. Both the locomotives and the cars have hand and Westinghouse air brakes. Each train has 22 cars and one locomotive.

Port.—The port is situated in the gulf of Valencia, on the Sagunto beach, between the ports of Valencia and Castellón, 12 miles from the former and 25 from the latter.

There is a breakwater 750 m. (2,460 ft.) long, normal to the shore and another 250 m. (820 ft.) long, perpendicular to the first. Between these a safe anchorage is provided against the levant storms which are the only ones to be feared along these coasts. There are 120 m. still to be built to the second breakwater mentioned. However, without this, the loading and discharging is carried on with great security, even in times of severe



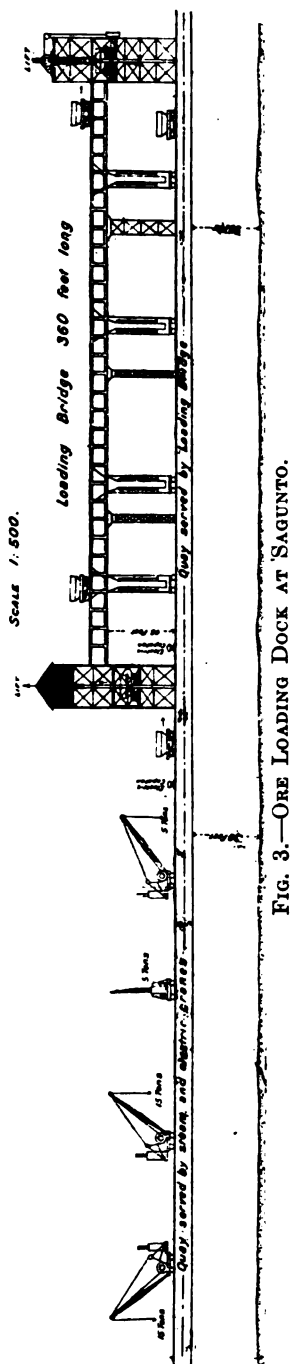


FIG. 3.—ORE LOADING DOCK AT SAGUNTO.

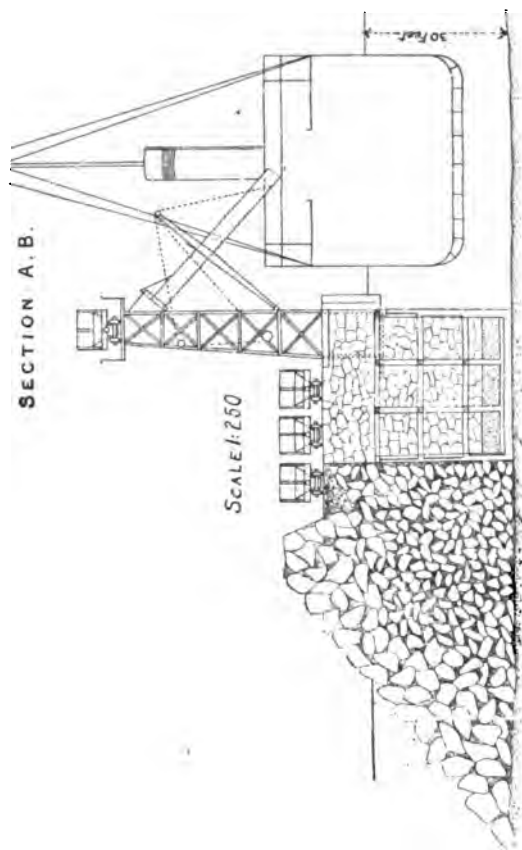


FIG. 4.—SECTION OF LOADING DOCK, FIG. 3.

storms. At present, two steamers 350 ft. long with a draft of 30 ft. can be loaded simultaneously. When the breakwater is completed it will be possible to work on a greater number of boats simultaneously. (See Fig. 2.)

The first 1,680 ft. of the first breakwater are of loose blocks; the remainder of the breakwater consists of eight concrete caissons weighing 5,150 tons each. On these caissons a large iron structure, similar to the equipment in the American ore docks, 360 ft. long and 46 ft. high, has been built. At one end of this dock, the railway trucks, 20 tons each, are elevated by a powerful electric lift, to the loading bridge, are run over this bridge and discharged through four chutes into steamers. The empty cars are lowered to the railway line by another elevator at the end of the dock.

The rest of the berthing quay is served by six electric and steam cranes. Two of these cranes are of 15 tons capacity each, and the other four are of 5 tons capacity. (See Figs. 3 and 4.)

The plant for loading and discharging has a capacity of 600 tons per hour. The port has the necessary electric lights for night illumination so that if necessary a night shift may be used. The port is of sufficient depth over a space of 39 acres to maneuver steamers of 30-ft. draft. Dredging is in progress which will increase this greatly. The stock piles at the port are capable of storing 100,000 tons of iron ore. This reduces to a minimum the delay in loading.

On the shore near the loading berths the company has two nodulizing plants, capable of producing 50 and 250 tons of iron-ore nodules, respectively, per day; also a briquetting plant consisting of 17 channel kilns, each producing 65 tons of iron-ore briquets per day, or 1,105 tons of briquets per day per plant.

The port expenses at Sagunto are light; the charge of 10 centimos per ton on the ore loaded includes the use of shore mooring ropes, mooring boat hands, pier pilot and of the quay. In addition, a transport tax is levied of 50 centimos per ton of iron ore shipped from Spain to the United Kingdom or to the Continent; to America the tax is only 20 centimos per ton.

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The Use of Low-grade Phosphates

BY JAMES A. BARR, B. S., E. M.,* MT. PLEASANT, TENN.

(New York Meeting, February, 1916)

WHEN phosphate mining operations first commenced in Tennessee the loss of both high- and low-grade material was large, because of the crude hand methods employed. Practically all rock smaller than 2 in. was thrown back into the pits and covered with the overburden of subsequent openings.

With the advent of mechanical washers and other mining machinery, together with the abolishing of the contract system, the waste has been steadily diminished, until at present only the finest sand is being rejected; and even this is not lost, for in the majority of cases the water, slime, and sand from the washers are run into large settling ponds where practically all of the phosphate-bearing material is caught and stored for future use, or until it is profitable to ship lower grades.

The latest equipment used to recover the fine and lower-grade phosphate sand includes Dorr thickeners, settling tanks, Dorr classifiers and settling ponds served by locomotive cranes with clam-shell buckets, making it possible to save material averaging as fine as 200 mesh. To attempt saving finer material would necessitate a different type of drier, which might be patterned after the multiple-deck roasters.

The present tailing runs from 15 to 25 per cent. of available phosphate, which, however, as mentioned before, does not represent a total loss to possible future operations. The demand of the consumer is the greatest factor in regulating the recovery at present.

In the mining of blue rock, which does not require washing treatment, the main waste material is the capping of low-grade kidney formation, generally used for backfilling of the rooms.

By reason of the introduction of improved methods in the early stages and the re-mining of territory skimmed over at the start, at the present time the net result of all operations is a comparatively small loss.

One of the largest temporary losses in the industry occurs in the Florida pebble district where the recovery of phosphate runs about 25 per cent. The loss is classed as temporary because most of the tailing has been

* Engineer, International Agricultural Corporation.

collected in ponds and piles, where it may be reclaimed in the future should developments or improved processes warrant.

In cleaning the Florida pebble phosphate, aside from the elutriation of the clayey binder and strata that sometimes contaminate the rock, the main operation is that of sizing, by which the large phosphate grains are separated from the smaller silica sand. This results in the loss of all the fine phosphate along with the silica. Owing to the similarity of the two in specific gravity, no mechanical process has been developed as yet to effect a further saving.

As the problem now stands, the development of some concentration process is required to effect a further recovery of the low-grade phosphate in Tennessee and Florida, where the consumers demand a high-grade product.

Taking 68 per cent. tricalcium phosphate as the present minimum average grade for Florida pebble and Tennessee blue rock, and 72 per cent. for the Tennessee brown rock, it will be seen that the development of such a concentration process is a much needed step in the conservation of the phosphate resources, for by its introduction the commercial limits for use in complete fertilizers could be reduced to 60 per cent. and possibly 50 per cent. tricalcium phosphate.

In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mt. Pleasant, Tenn., alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account. These figures are not claimed to be precise, but are given to indicate that the available phosphatic resources have, so to speak, merely been scratched.

It would seem that the most feasible means for making low-grade phosphate commercially available lies in the development of a smelting or electrometallurgical process whereby the phosphorus content of the rock would be rendered available (soluble) directly and without the introduction of a diluent, as in the manufacture of acid phosphate by the sulphuric acid method. By such a direct conversion, a 60 per cent. rock would serve the same purpose as a 72 to 75 per cent. rock with the sulphuric acid method.

Along these same lines a combination process may be developed whereby low-grade phosphate and an insoluble potash feldspar or similar potash mineral may, by heat or other treatment, be rendered available for use as a commercial fertilizer. The European war has brought out the importance of such a discovery.

Again, there might be developed another combination process whereby ammonia phosphate could be made directly, as cyanamid is now manufactured with the aid of an electric arc.

The greatest possibility of using low-grade phosphate lies in the direct application of finely ground rock to the soil in the raw state, allow-

ing the acids of nature to convert it into soluble forms. The limiting factor in this case is the freight charge on the inert and nonphosphatic portion.

The difficulty with the ground phosphate trade has been that the fineness of grinding was not such that a sufficient percentage of the commercial product became available within any reasonable time. When the grinding industry was new, the fineness was generally 90 per cent. through 80 mesh, while now it is 90 per cent. through 100 mesh, or even finer. The impalpable powder gives effective and immediate results; therefore the future extension of this particular phase of the phosphate industry will be aided greatly by finer grinding.

Another available source of phosphorus is basic slag from steel manufacture. The United States Steel Corporation has erected a plant in Alabama to prepare such slag for fertilizing purposes and has placed the product on the market. This form of fertilizer has been successfully used in Germany.

An average grain crop requires and takes about 15 lb. of phosphorus per acre from the soil in which it is grown. This phosphorus must eventually be replenished, for with a meager amount of phosphorus in the soil the crops are poor, and with none the harvest is nil. The majority of farmers fail to replenish the soil with the elements taken out by the crops, and at no very distant time they will be drawing heavily on our phosphate resources. In view of this, the use of the lower grades is certain to become common.

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The Iron Ores of the Philippine Islands*

BY WALLACE E. PRATT, A. M., E. M.,† MANILA, P. I.

(New York Meeting, February, 1916)

INTRODUCTION

IRON-ORE deposits in the Philippine Islands became the subject of official record as early as 1664. Undoubtedly iron ore was known and recognized by the Filipinos long before the earliest Spanish records. It is even possible that the Filipino practice of smelting iron ore is, like the native copper smelting, older than the Spanish conquest of the islands. The first records have to do with the magnetite-hematite ores near the towns of Santa Inez and Bosoboso in Rizal Province, from 15 to 20 miles east of Manila. The magnetite-hematite ores of Bulacan were exploited in 1783. Similar ores near the town of Mambulao in Camarines Province yielded a number of specimens to be exhibited in Spain in 1834. With the exception, then, of the lateritic iron ores on the eastern coast of northern Surigao, the true nature of which was not recognized until 1914, the important iron-ore deposits of the Philippines had all been discovered before the islands came under the dominion of the United States.

Indeed, in the Spanish documents relating to the history of iron mining are statements which, if their accuracy be not questioned, leave us in the Philippines today with much less iron ore than the Spaniards found. To quote, for example, a report submitted to the Superior Government in Spain in 1835 by one of its emissaries, Don Lorenzo Calvo:

"The Philippine Islands, most excellent Sir, possess iron and exhibit it in nearly every known mineralogical combination, and in such abundance that the cordillera of the island of Luzon, from Montufar Point in San Bernardino Straits to Cape Bojeador, is composed of no other mineral than iron, its ranges, of the third order, being in content and structure totally iron of the best quality, the more valuable in that its combinations are such as to facilitate its fusion and to render its reduction to metal simple."

Today, however, it is recognized that the iron-ore deposits in the Eastern Cordillera of Luzon constitute, not a continuous belt, but a

* Published by permission of the Director, Bureau of Science, Manila, P. I.

† Chief, Division of Mines, Bureau of Science.

series of widely separated deposits, including a half dozen orebodies within a distance of 10 miles in Bulacan Province, a single outcrop at

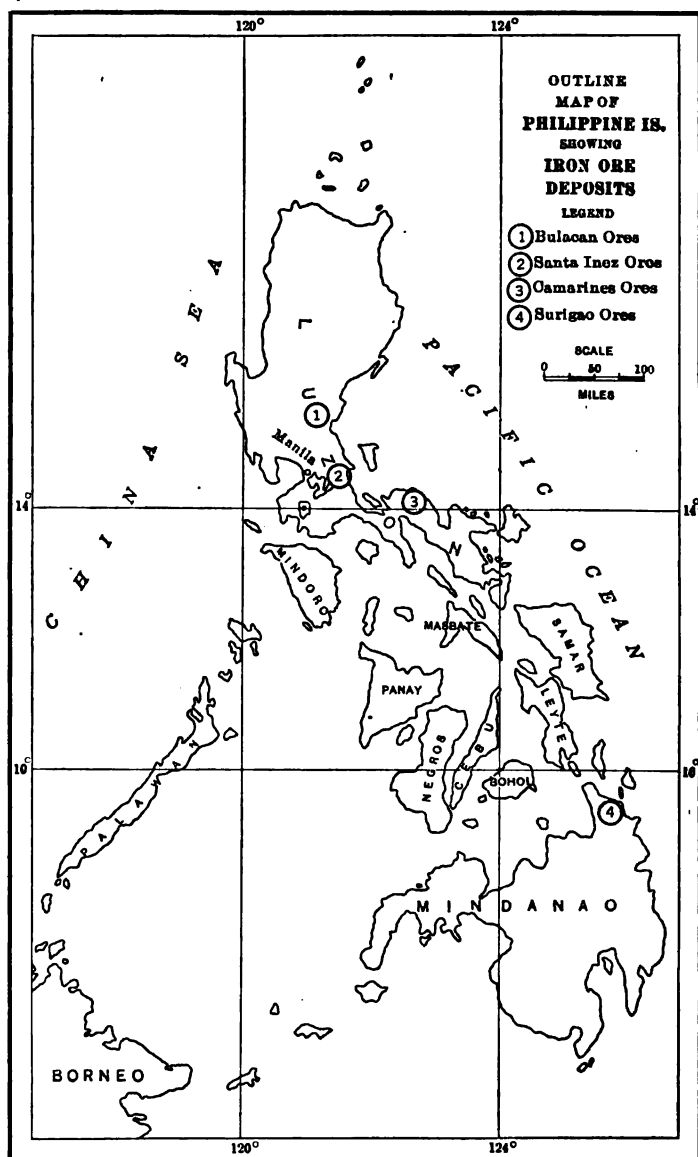


FIG. 1.

Santa Inez, Rizal Province, farther south, and three deposits around the margin of the Mambulao-Paracale gold-mining district in Camarines Province, 100 miles farther to the east-southeast. Other unimportant

occurrences of magnetite-hematite ores are known, but the foregoing, together with the lateritic ores in Surigao Province, Mindanao, contain the economically important iron-ore reserves of the Philippines.¹

MAGNETITE-HEMATITE ORES

General

The deposits of magnetite-hematite ores are found in the lower part near the base of the Miocene sedimentaries and along the contact of this series with the older igneous-complex upon which it rests. Some of the orebodies are entirely within the older igneous rocks close to the contact. Intrusive rocks are associated with each of the orebodies and penetrate the sedimentaries, the older igneous rocks, and even the orebodies at places. The igneous-complex includes deep-seated, intrusive, and extrusive types; in Bulacan, intrusive diorite and andesite, andesite flows, agglomerates, and tuffs; in Camarines, peridotite and pyroxenite. Both in Bulacan and in Camarines, also, there are areas of hornblende-granite which are older than the sedimentaries. In Camarines the granite has been rendered gneissic and the peridotite schistose, probably by regional dynamism. The sedimentary column is of varying thickness, but usually exceeds 3,000 ft.; at the base are breccias, conglomerates, and volcanic tuffs inclosing crystalline limestone; these are overlain by shale, subordinate sandstone, and tuff-sandstone, with two other limestone horizons in the upper part of the series.

Although outcrops of intrusive rocks are abundant near the orebodies, the exposures are small in area, discontinuous, and difficult to delineate. The predominant type of intrusive is felsitic, porphyritic, or holocrystalline in texture and subsiliceous in composition; the rock is dark colored and contains plagioclase feldspar and hornblende or, less commonly, pyroxene as essential minerals. Near one of the orebodies in Bulacan

¹ The following references on Philippine iron ores contain descriptions of the various deposits:

Dr. F. Rinne: *Zeitschrift für Praktische Geologie*, vol. x, p. 113 (1902).

Hiram Dryer McCaskey: A Geological Reconnaissance of the Iron Ores of Bulacan, *Bulletin of the Philippine Islands Mining Bureau*, No. 3 (1903).

G. I. Adams: Geological Reconnaissance of Southwestern Luzon, *Philippine Journal of Science*, vol. v, p. 106 (1910); contains notes on the Santa Inez, Rizal, ore.

F. A. Dalburg and Wallace E. Pratt: The Iron Ores of Bulacan Province, P. I., *Idem*, vol. iii, Sec. A, p. 201 (1914).

F. T. Eddingfield: A Microscopic Study of the Bulacan Iron Ores, *Idem*, vol. iii, Sec. A, p. 263 (1914).

Wallace E. Pratt and Victor E. Lednický: Iron Ore in Surigao Province, P. I., *Idem*, vol. v, Sec. A, p. 335 (1915).

Wallace E. Pratt: Iron Ore on Calambayanga Island, Mambulao, Camarines Province, P. I., *Idem*, vol. v, Sec. A, p. 323 (1915).

a siliceous rock is encountered in apparently intrusive relations; in this rock, quartz and feldspar phenocrysts, much cracked and with corroded or ragged outlines, are embedded in a microcrystalline groundmass of quartz and feldspar. The feldspar, both in the phenocrysts and in the groundmass, is principally orthoclase, but plagioclase feldspar is also present.

The walls of the ore deposits in Bulacan, for a distance equal, roughly, to the thickness of the orebody, consist almost invariably of an altered green rock of uncertain composition. So constant is the association of this wall rock with the ore that the relation is recognized by the native miners, who designate the greenstone as *camisa de bacal* (shirt or cloak of the iron ore). The wall rock is made up principally of amphibole and pyroxene with varying proportions of chlorite and epidote and often some plagioclase feldspar. Frequently the amphibole is fibrous and at many places hand specimens may be secured which consist entirely of either finely or coarsely fibrous amphibole. Eddingfield² identified the finely fibrous mineral in several specimens as tremolite. Adjacent to the ore the wall rock contains magnetite in grains and crystals of pyrite, disseminated and in veinlets with quartz. The apparent similarity of the wall rock at the Bulacan deposits to the "skarn" associated with the Scandinavian iron ores has been suggested in descriptions of the Bulacan ores.

The orebodies usually conform in strike and dip with the sedimentary formation and consist in some cases of replacements of sedimentary beds, especially limestone. Igneous rocks have also been replaced, however, and open fissures and cavities of irregular shape have been filled with ore. The surface dimensions of the outcrops are usually not great, the longest continuous exposure being something over 1,500 ft. in length. The orebodies have not been explored beneath the surface and some of the outcrops consist simply of blocks of ore scattered over the surface and embedded in clay.

The ore consists of magnetite and hematite in intimate mixture, but in proportions which vary at different outcrops. Magnetite, which is generally predominant, occurs as massive grains and the magnetite ores are usually soft. Hematite may be present only in negligible proportions or may constitute an ore to the practical exclusion of the magnetite. The hematite ores are harder than the magnetite and form resistant outcrops. The hematite is generally massive, but specularite occurs sparingly in nearly all of the deposits. Quartz is an abundant gangue mineral in the leaner ores and near the walls of the orebodies. It occurs in the interstices between the grains of magnetite and hematite. Eddingfield's microscopic work revealed needles of hematite penetrating the quartz in radiating groups and led him to the conclusion that the two

² *Loc. cit.*, p. 266.

minerals crystallized from the same solution. Quartz occurs, also, in secondary relations cutting the ore in small veins. A great deal of the country in the vicinity of the orebodies, likewise, is thoroughly silicified. Amphibole, chlorite and epidote are found in the ore near the walls. Pyrite occurs as grains and crystals in the ore as a primary constituent (according to Eddingfield) and as veinlets of secondary origin in the ore and walls. Pyrite is so abundant at places as to be objectionable in smelting but much of the ore, also, is free from pyrite. Rarely, chalcopyrite is found with the pyrite.

The geology of the iron ores will be made clearer, perhaps, by the following brief descriptions of the principal orebodies.

Descriptions of the Principal Deposits

The largest outcrop of iron ore in Bulacan Province is on the western flank of the Eastern Cordillera about 50 miles north of Manila, in a mountainous, inaccessible, and uninhabited region to which the name Camaching is applied. Camaching, in common with the other ore deposits in Bulacan, is in a heavily forested, broken country which can be entered only along narrow mountain trails. The nearest towns are 10 miles distant, while the railroad is 5 miles farther away. The Camaching deposit is at the head of a small river on a steep westward slope at an elevation of about 1,000 ft. In approaching it from the west along the river, the visitor traverses an extensive series of bedded tuffs and flows which represent the lower part of the Miocene sedimentaries. These beds strike north 20° east and dip to the westward at an angle of about 45°. The iron ore is encountered between the usual greenstone walls, previously described, near the base of the sedimentaries and the deposit conforms with the sedimentaries in strike and dip. The altered wall rock overlying the ore grades into fragmental bedded rocks containing a large proportion of volcanic tuff. Pieces of float ore downstream from the outcrop preserve the texture of the fragmental rock which the ore has replaced. In the wall of a stream crossing the outcrop an andesite dike, which follows the strike of the orebody, is exposed. The dike is about 15 ft. wide and is inclosed on both sides by ore. Similar dikes are encountered in the bedded rocks overlying the ore. Rounded blocks of crystalline limestone are included in the ore at places and upstream (stratigraphically beneath the orebody) is white crystalline limestone cut by small veins of magnetite. Blocks of calcareous, red hematite, evidently replacements of limestone, are found nearby. Farther up the slope sandy shale and tuff, which underlie the limestone, outcrop and, finally, at the top of the ridge are andesite flows which extend to the eastward for several miles.

The orebody as revealed by its outcrop is 1,500 ft. or more in length

and from 60 to 200 ft. wide in different exposures. The ore is principally magnetite, soft and massive. Quartz and pyrite occur in the ore and in the walls in the usual relations as previously set forth.

Another deposit of iron ore is found at Montamorong, about 4 miles south-southwest of Camaching. The orebody at this place lies on the eastern margin of a lens-shaped area of granite, elongated in a north-south direction but not exposed as far north as Camaching. The Miocene sedimentary series rests directly upon this granite along its western border and is found in patches on the eastern side of the granite, also. Detritus from the granite can be found in the lower beds of the sedimentaries, a condition which establishes the relative age of the two formations. For several miles to the east of the orebody at Montamorong the rocks are so thoroughly silicified as to be classified with difficulty but probably are tuffs and effusives of greater age than the ore deposit. In the vicinity of the orebody there are a number of exposures of unaltered subsiliceous rocks which are believed to represent intrusions, although the contact relations are not clear. The ore deposit, as revealed by a shallow pit, is a tabular body of magnetite from 3 to 7 ft. in thickness. The strike appears to be northwest and the pitch northeast, at an angle of 45°. The ore is soft and massive and contains quartz and considerable pyrite. The greenstone walls, likewise, carry much quartz and pyrite.

About 6 miles south-southwest of Camaching and in line with the strike of the orebody at that place, there are three outcrops of iron ore within a distance of $\frac{1}{2}$ mile, along a north-northeast line. These outcrops are exposed in the cañons of small streams and while the ore cannot be traced across the intervening ridges and may not persist from one orebody to another, yet the deposits are probably on the same structural line and related in origin. The outcrops are near the eastern margin of the granite area and are themselves flanked on the east by discontinuous exposures of sedimentaries. The sedimentary beds consist of schistose mottled limestone, schistose black shale, altered green tuff, and sandstone. The strike is uniformly north 15° east, parallel to the strike of the orebodies, and the dips are steep but are reversed in direction in different exposures. There are intrusive rocks in the immediate vicinity cutting both the granite and the sedimentaries. The quartz-bearing intrusive previously described is exposed between the middle and southern orebodies. Intrusives of subsiliceous composition are also present but at no place are the relations of the intrusives to the ore deposits clear.

The northernmost of the three outcrops is on the Constancia mining claim. The outcrop strikes north 15° east and the ore is apparently about 4 ft. thick, although it is stated that former exploration revealed a thickness of about 15 ft. The ore is magnetite with considerable py-

rite but very little quartz. In the hanging wall magnetite occurs in closely disseminated grains enmeshed in finely fibrous amphibole.

The Hison outcrop, 1,600 ft. south-southwest of Constancia, is opened by a shallow pit, 15 ft. wide and 10 ft. deep. The floor of the pit is entirely in magnetite and magnetite is exposed in the eastern and southern walls; blocks of magnetite and hematite, likewise, cover the hillside to the south. On the west, the pit is bounded by one wall of the deposit, striking north 20° east. The green, altered wall rock is particularly well exposed here, and the ore adjacent to the wall is contaminated slightly with amphibole and chlorite. The ore is free from pyrite and contains but little quartz although both of these minerals are found in the walls; in one wall is a small vein of pyritiferous quartz which strikes north 20° east.

About 1,000 ft. south-southwest of Hison is the third outcrop, called Santa Lutgarda. It has a tabular form, 15 ft. in thickness, and strikes north 15° east with a pitch of 60° to the west. The altered material of the foot wall grades into greenish tuff. The ore is hematite with but very little magnetite and contains more quartz than is present at the adjacent outcrops. In lean portions of the ore grains of massive hematite are set closely in a quartz groundmass. The hematite ore from Santa Lutgarda can be distinguished by a slight reddish tinge in contrast with the blue-black color of the magnetite ores. A small proportion of the Santa Lutgarda hematite is of the specular variety.

At Santol, a little more than 2 miles southwest of Santa Lutgarda, large blocks of hematite are encountered over an area 1,000 ft. long and 500 ft. wide on the lower slope of a hill. The blocks are embedded in residual clay and while some of them weigh, perhaps, 100 tons, no ore in place is to be seen. Santol is at the southern end of the granite exposure and the rocks beneath the hematite-strewn area are granite and andesitic intrusives, effusives, and tuffs in undetermined relation. Both on the east and west, Santol Hill is flanked by sedimentary rocks in which crystalline limestone is prominent. Blocks of the same limestone are found, also, mingled with the iron ore and lying just above the iron ore on the slope, but the direct replacement of limestone by iron ore, as observed at Camaching, is not evident at Santol. The ore is hematite, generally massive, with but little magnetite, and carries pyrite and much quartz. The blocks of ore appear to be arranged along a northeast line and iron-stained quartz can be traced to the northeast and to the southwest beyond the ends of the ore-strewn belt. The sedimentaries on either side of Santol strike north and dip west.

At Tumotulo, within 2 miles to the southwest of Santol, there is an insignificant quantity of iron ore which is interesting because, unlike the other Bulacan ores, it is titaniferous. The ore is found on the slope of a hill entirely within the sedimentary area. The hill is made up of shales

and is capped by limestone. Small pieces of ore, a massive magnetite-hematite mixture, together with pieces of porphyritic andesite are encountered on the surface immediately downhill from the limestone. Neither the ore nor the igneous rock is revealed in place. As shown by the analyses in Table I, the Tumotulo ore contains more than 9 per cent. of titanium.

There are several other places in Bulacan where small quantities of iron ore are obtained for smelting, but these minor deposits need not be described here.

The Santa Inez iron-ore deposit in Rizal Province, east of Manila, is in an area of sedimentary rocks intruded by andesite. The orebody is exposed in a small stream on the face of one of these intrusions. It is not apparent whether or not the ore penetrates the andesite. Crystalline limestone occurs farther upstream lying on the andesite and small veins of iron ore in limestone are to be observed in some of the loose blocks. The ore is principally hematite and is pyritiferous; chalcopyrite accompanies the pyrite in places. Blocks of hematite of many tons weight lie in the stream below the outcrop.

In Camarines Province iron ore is found in sedimentary rocks bordering an area of older peridotite and hornblende-granite. Associated with the iron-ore deposits in the base of the sedimentaries are numerous small intrusions of andesite, diorite, and gabbro. The largest outcrops of ore are on Calambayanga Island in Mambulao Bay and on the adjacent mainland. Mambulao Bay is a well-known harbor and there is deep water alongside Calambayanga Island so that the ore could be loaded onto ships at a minimum expense. The rocks inclosing the orebodies on the island and on the mainland near the island are steeply dipping, thin-bedded shales, sandstones, clastics, and conglomerates, with fragmentary exposures of crystalline limestone. The general strike of the beds and the trend of the orebody are north-northeast. Calambayanga Island is elongated in the same direction and a peninsula extends to the north-northeast from the mainland to within 2,000 ft. of the southern end of the island.

The island is $\frac{3}{4}$ mile long and half as much in width and blocks of iron ore, the average dimensions of which vary from 2 to 15 ft., are scattered profusely over its western half, from sea level up to an elevation of more than 200 ft. Along the beach the ore blocks are piled one on another but farther up on the slopes the individual blocks are embedded in the surface clay. Extending into the island from its southern extremity is a great body of quartz, 200 ft. wide and 75 ft. high. The quartz is iron-stained and intersecting veinlets of hematite cut through it in many places, while the sedimentary rocks adjacent to it are silicified. Toward the center of the island the quartz outcrop becomes concealed by a mantle of clay and on the northeastern point of the island where it

should appear, if it continued so far, large blocks of iron ore are piled high, one on another, over a belt 100 ft., or more, wide. Shales and sandstones, dipping at high angles, outcrop on either side of this ore but several yards to the south along the western shore of the island is a conspicuous dike, 20 ft. in width. The dike rock is holocrystalline in texture and consists essentially of plagioclase feldspar and decomposing pyroxene with accessory magnetite. South of the dike, which is vertical and strikes north 60° west, are volcanic tuff and breccia with blocks of ore again in great profusion 200 ft. farther south along the beach.

The ore is finely porous hematite with traces of fresh pyrite and quartz. It is moderately soft and the natural surfaces of the blocks are extremely pitted and irregular. Outcrops similar to those on the island occur on the point of the mainland nearby and reappear at intervals for a distance of more than a mile inland.

There is a deposit of magnetite at the village of Batobolani (the native word for magic stone, that is, magnetite) 7 miles southeast of Calambayanga Island. The ore occurs in large blocks embedded in yellowish-brown residual clay and covers an area about 1,200 ft. long by 600 ft. wide on the side of a hill. Dark-colored crystalline limestone and a subsiliceous holocrystalline rock which Dr. Rinne³ determined as hornblende-diorite, are found in loose pieces with the ore. The country at Batobolani is made up of intruded sedimentaries near the margin of the older peridotite-granite area.

A deposit of hematite, much of which is specular, of the same general character as the Batobolani occurrence but covering a smaller area, is found on the eastern border of the peridotite-granite formation about 12 miles east-southeast of Calambayanga Island.

Quality of the Magnetite-Hematite Ores

The chemical analyses in Table I afford an idea of the quality of the magnetite-hematite ores. These and other analyses show the total iron content to be usually more than 60 per cent. Silica is low in proportion to alumina except where the ore is quartzose, and magnesium oxide is noticeably high relatively to calcium oxide in several ores. Sulphur and phosphorus are usually not objectionably high. Attention has already been directed to the one titaniferous ore from Tumotulo, in Bulacan Province. Traces of cobalt have been detected in pyritiferous portions of some of the ores from Bulacan and it is a common experience to find fragments of cobalt-blue slag on the smelter dumps in that province. The analyses of Bulacan ore are upon samples taken from charges for the native smelters in several cases and probably show slightly more iron and less sulphur than the average ore contains.

³ *Loc. cit.*, p. 117.

TABLE I.—*Analyses of Magnetite and Hematite Ores from the Philippines*

Constituent	1	2	3	4	5	6	7
Moisture (100°C.).....	0.25	0.05		0.07	0.27	1.21	
Silica.....	5.02	4.66	43.76	6.84	3.32	9.52	1.02
Alumina.....	4.80	3.68	2.25	4.13	2.36	5.43	1.31
Ferric oxide.....	66.41	62.76	40.45	59.09	89.56	65.13	97.35
Ferrous oxide.....	20.64	27.32	13.50	27.68	4.13	7.89	
Lime.....	0.35	0.21	nil	0.20	0.16	0.38	
Magnesia.....	0.74	1.14		1.20	trace	0.58	
Manganese oxide.....	0.24	0.08		0.15	0.05	0.38	0.11
Titanium oxide.....	0.23	trace		0.20	trace	9.31	
Phosphorus pentoxide....	0.12	0.14		0.13	0.08	0.37	
Sulphur.....	0.02	0.21		0.21	0.06	0.01	
Carbon dioxide.....	1.10	0.30		0.30	0.18	0.53	
Total.....	100.32	100.50	99.96	100.13	99.90	99.53	99.79
Iron, metallic.....	62.54	65.17	38.80	61.71	65.91	51.72	64.14
Phosphorus.....	0.052	0.061		0.057	0.035	0.161	0.001

1. Camaching ore, smelter charge.
2. Hison ore, running sample across face 23 ft. long.
3. Quartzose ore from Hison.
4. Montamorong ore, smelter charge.
5. Santol ore, composite sample of spalls from many blocks.
6. Tumotulo ore, smelter charge.
7. Calambayanga ore, average composition of a 500-lb. sample.

Analyses 1 to 6, inclusive, by Forrest B. Beyer, formerly chemist Bureau of Science. Analysis 7 by T. Darjuan, chemist, Bureau of Science.

Genesis of the Magnetite-Hematite Ores

All the different deposits of magnetite and of hematite and of mixtures of these two minerals that have been described are believed to be related in origin. The features common to all of them are sufficient in number to justify this conclusion. The intimate association of the ore minerals with quartz and pyrite; the obvious replacement by ore of adjacent rocks, especially limestone, but, also, other types both sedimentary and igneous; the presence of intrusions near the orebodies; the character of the walls of the deposits; taken together, these observations point to the conclusion that the orebodies are the product of metamorphism resulting from intrusion, that is, metamorphism on or near the contact of intrusives with the intruded rocks. The presence of some of the typically contact minerals, like garnet, for instance, was nowhere detected but it may be suggested that, the intrusives being evidently limited in size, metamorphism may not have been extreme enough to have produced typical contact-metamorphic results. It is possible that hematite may have resulted at places from the oxidation of original

magnetite but it appears that hematite is also an original constituent with quartz in some of the ores. Pyrite is also a primary mineral according to Eddingfield.

Adams⁴ believes that the mineralization at Santa Inez resulted from the intrusion of andesite into sedimentary rocks. Rinne⁵ explained the Batobolani magnetite in Camarines as a product of contact metamorphism. C. M. Weld⁶ has described magnetite-hematite ores near Hongkong which are like the Philippine deposits in many respects and which he attributes to contact metamorphism resulting from the intrusion of the Hongkong granite into older sedimentaries. While the Hongkong granite is known to be of wide distribution and the Bulacan granite is similar to it in character, so that the correlation of the two rocks is not impossible, the Bulacan ores can not have resulted from the intrusion of the granite there into the sedimentaries, inasmuch as the granite is clearly older than the beds which have been replaced in part by iron ore. The intrusions involved in the mineralization in Bulacan are believed to be represented by the dike rocks, the presence of which has been noted. Dr. James F. Kemp has emphasized⁷ the striking similarity of the Bulacan iron ores as described by Dalburg and Pratt to contact-metamorphic iron ores at Daiquiri, Cuba.

Quantity of Magnetite-Hematite Ores

It will be obvious from what has been said that no accurate estimates of quantity can be made for the magnetite-hematite ores. Dalburg and Pratt,⁸ basing their estimate solely on outcrop dimensions and assuming that the orebody would continue in depth a distance equal to its least surface dimension, obtained 1,100,000 tons for the Camaching deposit. The other Bulacan deposits probably aggregate at least 100,000 tons. While the assumption as to the persistence of the ore in depth is conservative in view of the character of the mineralization believed to have caused these ore deposits, yet the data available are insufficient to make the estimates reliable. It is possible that the ore will contain a greater proportion of pyrite in depth and, consequently, should not be included in estimates of tonnage, but there is no indication on the surface that pyrite actually increases in depth. Removal of pyrite by oxidation near the surface, which would result in a proportionally greater pyrite content below the oxidized zone, does not appear to have taken place to any important extent. For the deposits of ore in

⁴ *Loc. cit.*, p. 106.

⁵ *Loc. cit.*, p. 113.

⁶ *Trans.* vol. 1, 236 to 245 (1914).

⁷ *Bulletin* No. 105, p. 1836 (September, 1915).

⁸ *Loc. cit.*, p. 235.

Rizal and Camarines no estimate of quantity can be offered. The Calambayanga deposit appears to be large; probably 100,000 tons are represented in the blocks of ore on the surface.

LATERITIC IRON ORES

General

On the eastern coast of Surigao Province in Mindanao there is an area of about 40 square miles, bordering the sea for a distance of 10

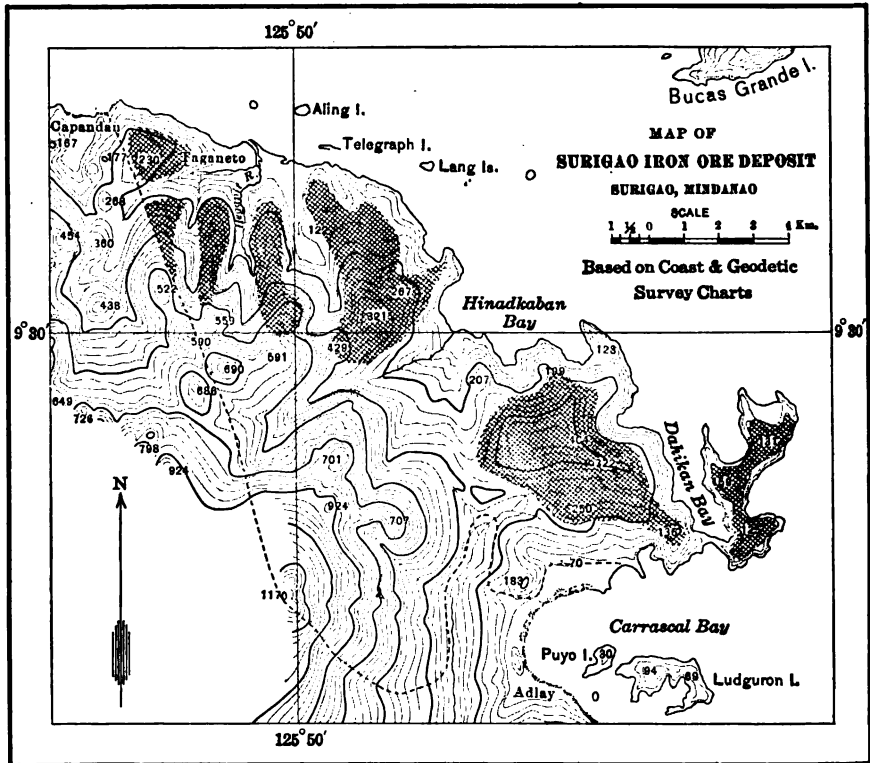


FIG. 2.—TOPOGRAPHIC MAP OF IRON-ORE DEPOSIT IN SURIGAO. CROSS-HATCHED AREAS BEST PARTS OF DEPOSIT. BROKEN LINE SHOWS APPROXIMATELY INTERIOR LIMIT OF DEPOSIT. ELEVATIONS IN METERS.

miles, which is conspicuously barren of vegetation and is covered with a bright red soil. This condition is the more notable in that the surrounding country is heavily forested and the singular barren appearance of this one section of the coast line has been a subject of interest for years to those who knew of it. But the region is sparsely inhabited and but few boats pass along that coast; consequently it was not until the year 1914

when H. F. Cameron, an engineer who is familiar with the iron ores of the Nipe Bay region in Cuba, made an inspection trip through Surigao that the "Red Hills," as the area in question has been designated on the Coast and Geodetic charts, were found to be covered with laterite rich enough in iron to constitute an ore. The first analyses of the ore were made in the Bureau of Science at Manila and a preliminary survey of the deposit has been made by the Division of Mines, Bureau of Science.

The laterite mantles a dissected plateau ranging up to 1,500 ft. in elevation and terminating along the coast in sea cliffs and steep slopes. This part of Mindanao receives a rainfall that averages about 10 ft. annually. Numerous small streams drain the area covered by laterite and, owing to the copious run-off have eroded deep cañons along their courses. The iron-ore deposit is thus separated into a number of small tracts by precipitous valleys on the walls of which no ore remains (Fig. 2). A peculiarity of the scant, shrub-like vegetation encountered over the laterite region is that most of the plants are common only to high altitudes in the Philippines, several of them not being found elsewhere below an elevation of 5,000 ft., whereas, here, they grow within a few hundred feet of sea level.

While there are no regular harbors near the ore deposit, Dahikan Bay, at its southern extremity, is almost landlocked and is perfectly protected in all seasons; its entrance is narrow but the water is deep, ranging from 18 to 28 fathoms within 300 ft. of the beach, and the inner bay is abundantly large. Two of the streams that reach the coast within the area covered by iron ore are large enough to afford considerable power, although it is not possible to present a definite estimate in this connection.

Character of the Iron Ores

The ore deposit and the ore itself are strikingly similar to the lateritic iron ores of the Mayari district in Cuba, as described by Kemp⁹ and by Leith and Mead¹⁰ before the Institute.

The Philippine laterite, or iron ore, is a surface blanket of residual clay varying in thickness up to 60 ft. and resting upon the parent rock from which it has been derived by tropical weathering processes. The surface and upper part of this mantle of hydrous iron oxides are brilliant reddish brown in color but at a depth of a few feet the color fades to a yellowish brown that persists downward to a point where the decomposition of the underlying rock is incomplete. In this zone of incomplete decomposition, which extends upward only 2 or 3 ft. from the hard, unchanged rock, the

⁹ James F. Kemp: The Mayari Iron-ore Deposits, Cuba, *Trans.*, vol. li, pp. 3 to 30 (1915).

¹⁰ C. K. Leith and W. J. Mead: Origin of the Iron Ores of Central and North-eastern Cuba, *Trans.*, vol. xlii, pp. 90 to 102 (1911).

color is pale green or almost white. The ore throughout its thickness is a spongy or mealy clay, but over its upper surface there are abundant small red concretions, together with occasional rusty porous fragments or crusts of the parent rock.

The parent rock, wherever exposed within the limits of the ore deposit, is essentially serpentine. Along the beach bordering the deposit there are many outcrops which uniformly represent rocks of the sub-siliceous class although the textures vary from preponderating holocrystalline to porphyritic, or even felsitic. No petrographic studies have been made of the specimens collected from these outcrops, but the most prominent rock may be classed megascopically as peridotite; probably, also, the porphyries and felsites, which are generally less completely decomposed than the holocrystalline rocks, represent small intrusive bodies.

Miocene sedimentaries, tuff-sandstones and limestone, abut upon the area of igneous rock covered by the ore and the contact between the two formations marks the interior limit of the ore deposit.

Composition of the Lateritic Iron Ore

In composition the Surigao iron ore appears to be slightly inferior to the ores of the same character in Cuba. The Surigao deposit was sampled by drilling, the material from each 10 ft. of hole constituting a separate sample. Thirty-one analyses made in the Bureau of Science upon material selected as representative of a total of 183 samples, taken at various depths from 89 different drill holes distributed throughout the ore deposit, show the average dry ore to contain 12.87 per cent. water and 47.40 per cent. metallic iron. Ore of this character would necessarily be sintered or nodulized before smelting and through the sintering process it would lose part of its water. Kemp¹¹ observes that the Cuban ore after being sintered contains from 3 to 3.5 per cent. of water. If the water in the Surigao ore is reduced to 3.5 per cent. by sintering prior to blast-furnace treatment the total iron content becomes 52.5 per cent., a figure which may be compared directly with 55.5 per cent. for the average sintered Cuban ore. If two conspicuously poor samples, both of which are from the bottoms of drill holes and are probably; therefore, contaminated with the underlying rock, be excluded from the 31 samples chosen as representative, the average iron content of Surigao ore, containing 3.5 per cent. of water, is 53.9 per cent.

The results of the sampling tests indicate that the Surigao ore is remarkably uniform in composition throughout the deposit. Of the 31 samples analyzed, 28 show more than 45 per cent., 24 show more than 50 per cent., 12 show more than 55 per cent., and three show more than

¹¹ James F. Kemp: *loc. cit.*, p. 131.

60 per cent. of metallic iron on the basis of 3.5 per cent. water content. Of the three samples containing less than 45 per cent. of iron on this basis, two samples might reasonably be discarded because of probable contamination as already explained. On the same basis the average iron content at different depths in five holes that are about 30 ft. each in depth is: upper 10 ft., 52.8 per cent.; from 10 to 20 ft. depth, 53.4 per cent.; and from 20 to 30 ft., 48.2 per cent., or 52.2 per cent. if one of the poor samples previously mentioned be excluded. These figures indicate that there is a slight increase in the iron content below the surface, followed by a decrease near the underlying rock. A similar variation is shown in holes of greater depth. A like increase in the content of iron in the ore immediately below the surface is a feature of the Cuban deposits, also, according to the papers previously quoted.

The only detailed analysis of the Surigao ore was performed upon a sample taken from the surface by Mr. Cameron. This analysis follows:¹²

	Per Cent.
Hygroscopic water.....	13.50
Combined water.....	6.60
Silica.....	1.04
Alumina.....	10.56
Ferric oxide.....	66.80
Ferrous oxide.....	0.36
Chromium oxide.....	1.15
Nickel oxide.....	nil
Sulphur.....	trace
Phosphorus.....	trace
Total.....	100.01
Metallic iron, dried ore.....	54.29
Metallic iron, ore deprived of combined water.....	58.20

In its lack of nickel, as represented by the single analysis, the Surigao ore is different in an important respect from the Cuban ore.

Quantity of Lateritic Iron Ore

In attempting to ascertain what quantity of ore is present in the Surigao deposit and what percentage of iron is contained in the average ore, hand drills were employed to determine the thickness of the ore mantle at different places and to secure representative samples of the ore from different depths. The area covered by iron ore is 40 square miles; although it was decided after inspection that about 12 square miles of this area is too distant from the coast and too mountainous to be considered economically important at present, the remaining area is much too large to admit of close sampling in a preliminary study. Con-

¹² Analysis by Francisco Pena, Bureau of Science.

sequently, groups of drill holes were placed so as to sample thoroughly small tracts in different parts of the economically important area. In all, 98 drill holes were sunk, admittedly too small a number of tests to establish closely and beyond question the composition and thickness of the ore mantle over an area of 28 square miles; nevertheless, the results obtained are so uniform even in widely separated parts of the deposit that they are believed to indicate, at least roughly, the tonnage and quality of the ore.

Of the 28 square miles of economically important area, only 12 square miles proved to be covered with ore of important thickness. The remaining portions of the deposit are either partly denuded because of the steepness of the slopes or are made up of alluvial-filled valleys in which the ore is contaminated with sand and gravel derived from the parent rock. Drill holes were placed so as to test four widely separated sections of the areas over which the ore mantle is relatively thick and one section in which the ore is thin and discontinuous, the holes in all cases being located at regular intervals, generally at the corners of 1,000-ft. squares. Of the holes located in areas over which the ore is of good depth, 6.8 per cent. fell upon bare places and encountered no ore; 28.8 per cent. encountered from 2 to 10 ft. of ore; 34.3 per cent. encountered from 10 to 20 ft. of ore; 21.9 per cent. encountered from 20 to 30 ft. of ore; 6.8 per cent. encountered from 30 to 40 ft. of ore; and 1.4 per cent. encountered from 40 to 50 ft. of ore. Of the holes in the poor area, 29.4 per cent. encountered no ore; 58.8 per cent. encountered from 2 to 10 ft. of ore; 5.9 per cent. encountered from 10 to 20 ft. of ore; and 5.9 per cent. encountered from 20 to 30 ft. of ore.

From these figures and the weight of a unit volume of the ore in place it can be calculated that the total economically important metric tonnage is 430,000,000, of which about 375,000,000 tons is contained in that part of the ore mantle which is 10 ft. or more in thickness. Reasonably accessible from the coast, but divided into a number of separate deposits by precipitous valleys, there is 275,000,000 tons of ore, with 260,000,000 tons lying 10 ft. or more deep. Finally, within two areas of ore which could be mined from a base at Dahikan Bay, the most feasible harbor site, there is 138,000,000 tons, 130,000,000 tons of which is more than 10 ft. thick. It should be noted, however, that the bulk of even this most favorably situated ore lies on the tops of hills and broad divides from 400 to 1,000 ft. above sea level and that within each of the two areas near Dahikan Bay there are ravines and denuded slopes which would of necessity be avoided in mining.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Development of the Law Relating to the Use of Gas Compressors in Natural Gas Production

BY SAMUEL S. WYER,* COLUMBUS, OHIO

(New York Meeting, February, 1916)

THE art of natural-gas compressing is now over 25 years old, and has grown at practically the same rate as the increase in domestic natural-gas consumers. There are now over 200 natural-gas compressing stations in North America, aggregating more than 320,000 hp. of compressor capacity and representing a property value of more than \$22,000,000, and compressing more than 85 per cent. of all the gas used. The age and magnitude of the art make it evident that the use of gas compressors is a recognized integral part and universal custom of the natural-gas business.

The public also is not without its rights and vital interests in this problem. Approximately 1,700,000 domestic natural-gas consumers in North America are dependent upon gas compressors for their natural-gas service. That is, if the use of compressors were to be prohibited, the majority of these consumers would be unable to secure adequate natural-gas service.

Each consumer represents between four and five persons, and it therefore follows that the comfort and well-being—as far as natural-gas service is concerned—of at least 8,000,000 persons would be affected if the use of natural-gas compressors should be prohibited. These 1,700,000 consumers have invested, in services, house piping, fixtures, and appliances, an average of about \$90 each, or an aggregate of \$153,000,000, which is much more than the companies' investment in gas compressors. Furthermore, in all cases where the rate paid by the consumers is fair to the gas company—considering the value of the service rendered by the gas company—the consumers are entitled to continued service and protection of the investment they have made on the faith that the gas service would be continued in the future.

Since "Customs adopted and acquiesced in, if not in conflict with federal or State legislation, have the force of positive law,"¹ and "Courts will take notice of whatever ought to be generally known within the limits of their jurisdiction,"² there ought to be no question as to the unqualified right to use gas compressors. However, many small gas-pro-

* Consulting Engineer.

¹ Lindley on Mines, Sec. 271, vol. i, 3rd ed.

ducers, not using gas compressors, have sought to secure permanent injunctions from courts to prevent other gas-producers from using such compressors. This has resulted in much expensive litigation. The following is a chronological arrangement of all the court decisions relating to this question, and shows the logical development of the common law. To expedite cross-reference work, each case is given a serial number; also, abbreviations used in citations are given in full in the accompanying table.

Ch. Div.....	Chancery Division, British Law Reports.
N. E.....	Northeastern Reporter.
Ind.....	Indiana Reports.
Fed. Rep.....	Federal Reporter.
U. S.....	United States Reports.
Pa.....	Pennsylvania State Reports.
Pac. Rep.....	Pacific Reporter.
S. W. Rep.....	Southwestern Reporter.
Ohio State.....	Ohio State Reports.
Oklahoma.....	Oklahoma Reports.

There are four classes of cases, namely, those that relate to:

Common-law rights in use of gas compressors. Nos. 1, 3, 7, 10, 12, 13, 26.

Rights under State laws regulating use of gas compressors. Nos. 4, 14, 16, 17, 19, 20.

Judicial recognition of declining gas volumes. Nos. 7, 8, 10, 12, 13, 21, 22, 23, 29.

Questions of "ownership," "possession," or "right of transportations" of natural gas. Nos. 2, 5, 6, 7, 8, 9, 11, 13, 15, 18, 21, 24, 25, 27, 28, 29, 30, 31.

1885

No. 1.—Ballard vs. Tomlinson, 29 Ch. Div. 122.

"The plaintiff, if he has a right to use anything in nature, has the right to exercise that use by all the skill and invention of which man is capable, and it seems to me that as long as the plaintiff uses only lawful means as against his neighbor, however ingenious and however artificial those means may be, his right to appropriate the common source is not diminished because he uses the most artificial or the most ingenious methods." Cited in:

No. 12.—Jones vs. Forest Oil Co., 194 Pa. 379.

1889

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., Supreme Court of Indiana, 22 N. E. 778.

"Natural gas is as much an article of commerce as iron ore, coal, petroleum, or any other of the like products of the earth. It is a commodity which may be transported, and it is an article which may be bought and sold in the markets of the country. * * * The power to regulate commerce between the States is exclusively in the federal congress. An action by congress will not authorize the States to legislate in matters of interstate commerce." * * * Transportation of commercial commodities

² U. S. Supreme Court, Brown vs. Spilman, 155 U. S., 670.

from State to State is interstate commerce, and the State legislatures can neither burden nor restrict it." Cited in:

- No. 4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.
- 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.
- 9.—Townsend vs. State, 147 Ind. 624.
- 15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.
- 18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.
- 19.—Richmond Natural Gas Co. vs. Enterprise Nat. Gas Co., Appellate Court of Indiana, 66 N. E. 782.
- 28.—Haskell vs. Cowham, 187 Fed. Rep. 403.
- 30.—West vs. Kansas Nat. Gas Co., 221 U. S. 229.

1889

No. 3.—Westmoreland, etc., Co. vs. DeWitt, Supreme Court of Pennsylvania, 130 Pa. 235.

"Gas, it is true, is a mineral; but it is a mineral with peculiar attributes, which require the application of precedents arising out of ordinary mineral rights, with much more careful consideration of the principles involved than of mere decisions. Water also is a mineral; but the decisions in ordinary cases of mining rights, etc., have never been held as unqualified precedents in regard to flowing, or even to percolating, waters. Water and oil, and still more strongly gas, may be classed by themselves, if the analogy be not too fanciful, as minerals *feræ naturæ*. In common with animals, and unlike other minerals, they have the power and the tendency to escape without the volition of the owner. Their 'fugitive and wandering existence within the limits of a particular tract was uncertain' * * * They belong to the owner of the land, and are a part of it, so long as they are on or in it, and are subject to his control; but when they escape, and go into other land, or come under another's control, the title of the former owner is gone. Possession of the land, therefore, is not necessarily possession of the gas. If an adjoining, or even a distant, owner drills his own land, and taps your gas, so that it comes into his well and under his control, it is no longer yours, but his * * * the one who controls the gas—has it in his grasp, so to speak—is the one who has possession in the legal as well as in the ordinary sense of the word." Cited in:

- No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.
- 8.—Brown vs. Spilman, 155 U. S. 665.
- 9.—Townsend vs. State, 147 Ind. 624.
- 12.—Jones vs. Forest Oil Co., 194 Pa. 379.
- 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.
- 25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

1891

No. 4.—Jamieson vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 128 Ind. 555.

"The Indiana statute prohibiting the transportation of natural gas through pipes at a greater pressure than 300 lb. per square inch, or otherwise than by its natural flow, is a valid exercise of the police power of the State, since natural gas is an intrinsically dangerous substance, and the legislature having determined what pressure is reasonable and safe the courts cannot review its action. * * * That natural gas is a dangerous agency is a matter of common knowledge; and hence courts take judicial notice of that fact. We know, as the legislature knew, and as every one knows, that natural gas is in a high degree inflammable and explosive. * * * It would be unreasonable to hold that the courts know judicially that natural gas is a public necessity so far as to warrant the exercise of the right of eminent domain, and yet hold that they do not know that it is inflammable and explosive. Knowing the one thing, they must

know the other. We hold without hesitation that natural gas is so dangerous that its use may be made the subject of a police regulation. Decision after decision recognizes the principle we have stated, and upholds laws regulating the use of property. * * * The public safety and welfare is the highest consideration in all legislation, and to this consideration private rights must yield. No man has a right to so use a dangerous species of property as to put the safety of others in peril. Liberty does not imply the right of one man to so use property as to endanger the property of others; nor does ownership imply any such right. This is rudimental. It must therefore be true that the owner of property of such a dangerous nature as to require regulation to prevent injury to others can have no right paramount to the police power. It is not too much to say that, as against the police power, there is no such thing as a vested right. * * * No investment, however great, can so vest a right as to preclude the just exercise of a great governmental power, such as that under which regulations for the protection of the health and safety of persons are enacted. This principle is supported by many decisions. * * * We have already declared that it is a dangerous substance, requiring regulation, and we shall only add to what we have said a quotation from the opinion. * * * 'It was not necessary,' said the court, 'to aver that coal-oil is inflammable, or to prove it. Courts and juries will take cognizance of such matters as are of common knowledge, and pertain to the experience and affairs of almost every man's daily life. Courts do not require proof that fire will burn, or powder explode, or gas illuminate, or that many other processes in nature and art produce certain known effects.'" Cited in:

No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

9.—Townsend vs. State, 147 Ind. 624.

15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

(It is important to note that Case No. 19 distinguishes between the use of gas compressors for transporting gas and for pumping wells.)

25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

30.—West vs. Kansas Natural Gas Co., 221 U. S. 229.

1892

No. 5.—People's Gas Co. vs. Tyner, Supreme Court of Indiana, 131 Ind. 277.

"It has been settled in this State that natural gas, when brought to the surface of the earth and placed in pipes for transportation, is property, and may be the subject of interstate commerce. Water, petroleum oil, and gas are generally classed by themselves as minerals possessing in some degree a kindred nature. As to whether the owner of the soil may dig down and divert a well-defined subterranean stream of water, there is much diversity of opinion and conflict in the adjudicated cases; but the authorities agree that the owner of a particular tract of land may sink a well and appropriate to his own use all the percolating water found therein, though it may entirely destroy the well on his neighbor's land. * * * It is a familiar maxim that in contemplation of law, land always extends downward as well as upward, so that whatever is in a direct line between the surface of any land and the center of the earth belongs to the owner of the surface. * * * When it is once conceded that the owner of the surface has the right to sink a well and draw gas from the lands of an adjoining owner, no valid reason can be given why he may not enlarge his well by the explosion of nitroglycerine therein for the purpose of increasing the flow. The question is not as to the quantity of gas he may take, but it is a question of his right to take the gas at all." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., Supreme Court of Indiana, 222 N. E. 778.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

Cited in:

- No. 6.—Tyner vs. People's Gas Co., 131 Ind. 408.
- 9.—Townsend vs. State, 147 Ind. 624.
- 11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.
- 18.—Federal Oil Co. vs. Western Oil Co., U. S. Circuit Court of Appeals, 121 Fed. Rep. 674.

1892

- No. 6.—Tyner vs. People's Gas Co. Supreme Court of Indiana, 131 Ind. 408.
This merely reaffirms the doctrine in the preceding case. Cites:
- No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

1893

- No. 7.—Hague vs. Wheeler, Supreme Court of Pennsylvania, 157 Pa. 324.

"A court of equity will not interfere by injunction to compel a landowner who has sunk a gas well on his own premises without malice or negligence to stop the flow of gas therefrom, which has proven insufficient in quantity to enable him to utilize it, at the suit of the adjoining owners, whose wells yield gas in sufficient quantities to enable them to utilize and market it, though defendant's well drains the common reservoir, and thus will ultimately reduce the flow of plaintiffs' wells. * * * The mere fact that the defendants, by operations upon their land, are taking gas from the earth, and thereby diminishing the quantity of gas which would otherwise come to the plaintiffs' wells, furnishes no ground for complaint or equitable interference." In speaking of oil and gas the Court said, "But they are not, like coal and iron ore, fixed in their place in the rocks, so that the owner may know his own, protract his lines downward to mark his boundaries, and take them when he pleases. As water percolates by untraceable rills through the gravel, so these 'minerals *feræ naturæ*,' as they have been aptly called in a recent case, permeate the porous rocks deep in the bowels of the earth, and rush to the surface through any opening made through the impervious cap by which the basin which contains them is sealed. No landowner gets through his wells oil or gas exclusively from his own land. That which saturates his rocks may be lawfully taken by his neighbor through wells on his land, tapping the common reservoir. From the very nature of the case, the right of each owner is qualified. It is common to all whose land overlies the basin, and each must of necessity exercise his right with some regard to the rights of the others. * * * The common law is a growing tree; its principles must be continually adapted to new facts, and the changing conditions of modern life. Only the legislature can grub it up, but the courts are charged with the duty of pruning its branches, and sometimes grafting a new scion on the old stock. * * * it is said that the oil and gas are unlike the solid minerals, since they move through the interstitial spaces or crevices in the sand rocks in search of an opening through which they may escape from the pressure to which they are subject. This is probably true. It is one of the contingencies to which this species of property is subject. But the owner of a surface is an owner downward to the center, until the underlying strata have been severed from the surface by sale. What is found within the boundaries of his tract belongs to him according to its nature. The air and water he may use. The coal and iron or other solid minerals he may mine and carry away. The oil and gas he may bring to the surface and sell in like manner, to be carried away and consumed. His dominion is, upon general principles, as absolute over the fluid as the solid minerals. * * * He cannot estimate the quantity in place of gas or oil, as he might of the solid minerals. He cannot prevent its movement away from him, toward an outlet on some other person's land, which may be more or less rapid, depending on the dip of the rock or the coarseness of the sand composing it; but so long as he can reach it and bring it to the surface it is his absolutely, to sell, to use, to

give away, or to squander, as in the case of his other property. In the disposition he may make of it he is subject to two limitations: he must not disregard his obligations to the public; he must not disregard his neighbor's rights. If he uses his product in such a manner as to violate any rule of public policy or any positive provision of the written law, he brings himself within the reach of the courts. If the use he makes of his own, or its waste, is injurious to the property or the health of others, such use or waste may be restrained, or damages recovered therefor; but, subject to these limitations, his power as an owner is absolute, until the legislature shall, in the interest of the public as consumers, restrict and regulate it by statute." Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

1895

No. 8.—Brown vs. Spilman, U. S. Supreme Court, 155 U. S. 665.

"Petroleum gas and oil are substances of a peculiar character, and decisions in ordinary cases of mining for coal and other minerals which have a fixed situs, cannot be applied to contracts concerning them without some qualifications. They belong to the owner of the land, and are part of it, so long as they are on it or in it or subject to his control; but when they escape and go into other land, or come under another's control, the title of the former owner is gone. If an adjoining owner drills his own land, and taps a deposit of oil or gas, extending under his neighbor's field, so that it comes into his well, it becomes his property. * * * When oil or gas is found in paying quantities, it is not usual to consume it or reduce it to use at the wells, but it is conducted in iron pipes to large tanks or reservoirs, whence it is distributed by other pipes to the places of consumption, often many miles distant. These are matters within the common experience or knowledge of all men living in those portions of the country where oil and gas are produced, and courts will take notice of whatever ought to be generally known within the limits of their jurisdiction." Cites:

No. 3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

22.—Brewster vs. Lanyon Zinc Co., U. S. Circuit Court of Appeals, 140 Fed. Rep. 801.

25.—Kansas Natural Gas Co. vs. Haskell, U. S. Circuit Court, 172 Fed. Rep. 545.

1897

No. 9.—Townsend vs. State, Supreme Court of Indiana, 147 Ind. 624.

Affirms State's right in preventing the burning of natural gas in flambeau lights on account of the wasteful use of natural gas in this manner, since the wasteful use of gas which is drawn from a common storage not reduced to individual possession is an injury to others. Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

Cited in:

No. 11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.

1897.

No. 10.—Thomas C. Kelly vs. Ohio Oil Co., Ohio Supreme Court, 57 Ohio State 317.

"When a person has the legal right to do a certain act, the motive with which it is done is immaterial. The right to acquire, enjoy, and own property carries with it the right to use it as the owner pleases, so long as such use does not interfere with the legal

rights of others. * * * Whatever gets into the well belongs to the owner of the well, no matter where it came from. In such cases the well and its contents belong to the owner or lessee of the land, and no one can tell to a certainty from whence the oil, gas, or water which enters the well came, and no legal right as to the same can be established or enforced by an adjoining landowner. The right to drill and produce oil on one's own land is absolute, and cannot be supervised or controlled by a court or an adjoining landowner. So long as the operations are legal, their reasonableness cannot be drawn in question. * * * Petroleum oil is a mineral, and while in the earth it is part of the reality, and, should it move from place to place by percolation or otherwise, it forms part of that tract of land in which it tarries for the time being, and, if it moves to the next adjoining tract, it becomes part and parcel of that tract; and it forms part of some tract until it reaches a well, and is raised to the surface, and then for the first time it becomes the subject of distinct ownership, separate from the realty, and becomes personal property—the property of the person into whose well it came. And this is so whether the oil moves, percolates, or exists in pools or deposits. In either event, it is the property of, and belongs to, the person who reaches it by means of a well, and severs it from the realty, and converts it into personalty. While it is generally supposed that oil is drained into wells for a distance of several hundred feet, the matter is somewhat uncertain, and no right of sufficient weight can be founded upon such uncertain supposition to overcome the well-known right which every man has to use his property as he pleases, so long as he does not interfere with the legal rights of others. * * * While the drilled oil well is artificial, the pores and channels through which the oil reached the bottom of the well are natural.” Cited in:

No. 25.—*Kansas Natural Gas Co. vs. Haskell*, 172 Fed. Rep. 545.

1898

No. 11.—*State of Indiana vs. Ohio Oil Co.*, Indiana Supreme Court, 150 Ind. 21.

“The title to natural gas does not vest in any private owner until it is reduced to actual possession. A statute making it unlawful to permit the escape of natural gas into the open air from a well for longer than two days after it is constructed is not unconstitutional. The continuous and persistent waste of natural gas to the detriment of the community at large and in violation of statute is a nuisance which may be abated by injunction. * * * In the light of these facts, one who recklessly, defiantly, persistently, and continuously wastes natural gas, and boldly declares his purpose to continue to do so, as the complaint charges appellee with doing, all of which it admits to be true by its demurrer, ought not to complain of being braided as the enemy of mankind. * * * It is not the use of unlimited quantities of gas that is prohibited, but it is the waste of it that is forbidden. The object and policy of that inhibition is to prevent, if possible, the exhaustion of the store house of nature, wherein is deposited an element that ministers more to the comfort, happiness, and well-being of society than any other of the bounties of the earth. * * * But we cannot have the blessing of natural gas unless the measures for the preservation thereof in this State are enforced against the lawless. We therefore conclude that the facts stated in the complaint make a case of public nuisance which the appellant has a right to have abated by injunction, and that the complaint states facts sufficient to constitute a cause of action.”

Affirmed by U. S. Supreme Court in:

No. 13.—*Ohio Oil Co. vs. Indiana*, 177 U. S. 190.

Cites:

No. 5.—*People's Gas Co. vs. Tyner*, 131 Ind. 277.

No. 9.—*Townsend vs. State*, 147 Ind. 624.

Cited in:

No. 25.—*Kansas Nat. Gas Co. vs. Haskell*, 172 Fed. Rep. 545.

1900

No. 12.—Jones vs. Forest Oil Co., Supreme Court of Pennsylvania, 194 Pa. 379.

"A gas pump may lawfully be used to increase the production of an oil well, although the production of wells on adjoining property is thereby diminished. * * * The question here is to what extent an owner of oil wells may use mechanical devices for bringing the oil to the surface. In operating his own wells, may he use appliances which diminish the production of his neighbor's wells? It is not denied that a gas pump will to some extent affect the production of oil wells located in the immediate neighborhood of the well to which the pump is attached, if the sand from which the oil is obtained is of a porous, pebbly nature, as is the case in the McCurdy field. * * * An owner of land may dig a well upon his property, and if, in so doing, he taps the hidden flow of water, which supplies his neighbor's spring, it is a loss to the neighbor for which the law provides no remedy. * * * 'It must be conceded, we think, that every man is entitled to the ordinary and natural use and enjoyment of his property; he may cut down the forest trees, clear and cultivate his land, although in so doing he may dry up the sources of his neighbor's springs, or remove the natural barriers against wind and storm. * * * In sinking his well he may intercept and appropriate the water which supplies his neighbor's well.' In this case the defendant has the exclusive right to bore for oil. * * * The right being a lawful one, the defendant is at liberty to use all lawful means to obtain all the gas and oil contained in, or obtainable through, the land.' * * * And to that end it may resort to the use of all known lawful modern machinery and appliances. The plaintiff's claim is that the use of a gas pump in the production of oil is unlawful, because, as he alleges, by its powerful suction the oil and gas are drawn from his adjoining farm, thereby decreasing his production. Plaintiff assumes that there is a certain fixed amount of oil and gas under his farm, in which he has an absolute property. True, they belong to him while they are a part of his land; but when they migrate to the lands of his neighbor, or become under his control, they belong to the neighbor. * * * The principle of natural user does not apply at all. The plaintiff, if he has a right to use anything in nature, has a right to exercise that use by all the skill and invention of which man is capable, and it seems to me that as long as the plaintiff uses only lawful means as against his neighbor, however ingenious or however artificial those means may be, his right to appropriate the common source is not diminished because he uses the most artificial or most ingenious methods.' If it is lawful to take water from a substrata by the "exercise of all the skill and invention of which man is capable," we see no reason why it is not lawful to produce oil by those means, especially as the possession of the soil for purposes of tillage gives the owner no actual possession of the oil and gas underlying it. The evidence shows that the gas pump has been in constant use in all fields, except one, to a greater or less extent, since the discovery of oil; that its use has been generally recognized by all operators; and that it is only used on wells in territory which is almost exhausted. * * * In view of the testimony and authorities above cited, we conclude that the use of a gas pump by defendant, under the circumstances of this case, is not an unlawful act that should be restrained by injunction; that the plaintiff is not entitled to the relief prayed for; and that the bill should be dismissed. Cites:

No. 1.—Ballard vs. Tomlinson, 29 Ch. Div. 194.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

1900

No. 13.—Ohio Oil Co. vs. Indiana, Supreme Court of the United States, 177 U. S. 190.

* * * "oil and gas, like other minerals, are situated beneath the surface of the earth, but except for this one point of similarity, in many other respects they greatly differ. They have no fixed situs under a particular portion of the earth's surface within the area where they obtain. They have the power, as it were, of self-transmission. No one owner of the surface of the earth, within the area beneath which the gas and oil move, can exercise his right to extract from the common reservoir, in which the supply is held, without, to an extent, diminishing the source of supply as to which all other owners of the surface must exercise their rights. * * * Now, it is doubtless true that the public has a sufficient interest in the preservation of oil and gas from waste to justify legislation upon this subject. Something has been done in this direction already by the acts regulating the plugging of abandoned wells. * * * In the disposition he may make of it (private property) he is subject to two limitations. He must not disregard his obligations to the public. He must not disregard his neighbor's rights. If he uses his product in such a manner as to violate any rule of public policy, or any positive provisions of the written law, he brings himself within the reach of the courts. If the use he makes of his own, or its waste, is injurious to the property or the health of others, such use or waste may be restrained, or damages recovered therefor; but, subject to these limitations, his power as an owner is absolute until the legislature shall, in the interest of the public, as consumers, restrict and regulate it by statute. * * * the rule of property in the state of Indiana to be as follows: Although in virtue of his proprietorship the owner of the surface may bore wells for the purpose of extracting natural gas and oil until these substances are actually reduced by him to possession, he has no title whatever to them as owner. That is, he has the exclusive right on his own land to seek to acquire them, but they do not become his property until the effort has resulted in dominion and control by actual possession. It is also clear from the Indiana cases cited that, in the absence of regulation by law, every owner of the surface within a gas field may prosecute his efforts and may reduce to possession all or every part, if possible, of the deposits, without violating the rights of the other surface owners. * * * But whilst there is an analogy between animals *feræ naturæ* and the moving deposits of oil and natural gas, there is not identity between them. Thus, the owner of the land has the exclusive right on his property to reduce the game therefound to possession, just as the owner of the soil has the exclusive right to reduce to possession the deposits of natural gas and oil found beneath the surface of his land. The owner of the soil cannot follow game when it passes from his property; so, also, the owner may not follow the natural gas when it shifts from beneath his own to the property of someone else within the gas field. It being true as to both animals *feræ naturæ* and gas and oil, therefore, that whilst the right to appropriate and become the owner exists, proprietorship does not take being until the particular subjects of the right become property by being reduced to actual possession. The identity, however, is for many reasons wanting. In things *feræ naturæ* all are endowed with the power of seeking to reduce a portion of the public property to the domain of private ownership by reducing them to possession. In the case of natural gas and oil no such right exists in the public. It is vested only in the owners in fee of the surface of the earth within the area of the gas field. * * * as to gas and oil the surface proprietors within the gas field all have the right to reduce to possession the gas and oil beneath. They could not be absolutely deprived of this right which belongs to them without a taking of private property. * * * In view of the fact that regulations of natural deposits of oil and gas and the right of the owner to take them as an incident of title in fee to the surface of the earth, as said by the Supreme Court of Indiana, is ultimately but the regulation of real property, and they must hence be treated as relating to the preservation and protection of rights of an essentially local character. Considering this fact and the peculiar situation of the substances, as well as the character of the rights of the surface owners, we cannot say that the statute amounts to a taking of private

property, when it is but a regulation by the State of Indiana of a subject which especially comes within its lawful authority." Affirms:

No. 11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.

Cites:

No. 3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

7.—Hague vs. Wheeler, 135 Pa. 324.

8.—Brown vs. Spilman, 185 U. S. 665.

12.—Jones vs. Forest Oil Co., 194 Pa. 379.

Cited in:

No. 18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.

19.—Richmond Natural Gas Co. vs. Enterprise Nat. Gas Co., Appellate Court of Ind., 66 N. E. 782.

22.—Brewster vs. Lanyon Zinc Co., 140 Fed. Rep. 801.

25.—Kansas Nat. Gas Co. vs. Haskell, 172 Fed. Rep. 545.

28.—Haskell vs. Cowham, 187 Fed. Rep. 403.

30.—West vs. Kansas Natural Gas Co., 221 U. S. 229.

1900.

No. 14.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 155 Ind. 461.

"The question to be determined here is not whether natural gas, when reduced to possession, is property, but as to the right of well owners to use certain extraordinary means to reduce it to possession. * * * Natural gas is a fluid mineral substance, subterranean in its origin and location, possessing in a restricted degree the properties of underground waters, and resembling water in some of its habits. Unlike water, it is not generally distributed, and, so far as now understood, it can be used for but few purposes; the most important being that of fuel. Its physical occurrence is in limited quantities only, within circumscribed areas of greater or less extent. * * * but the difference between natural gas and underground waters, whether flowing in channels or percolating the earth, is so marked that the principles which the courts apply to questions relating to the latter are not adapted to the adjustment of the difficulties arising from conflicting interests in this new and peculiar fluid. Natural gas, being confined within limited territorial areas, and being accessible only by means of wells or openings upon the lands underneath which it exists, is not the subject of public rights in the same sense or to the same extent as animals *feræ naturæ* and the like are said to be. Without the consent of the owner of the land, the public cannot appropriate it, use it, or enjoy any benefit whatever from it. * * * Natural gas in the ground is so far the subject of property rights in the owners of the superincumbent lands, that while each of them has the right to bore or mine for it on his own land, and to use such portion of it as, when left to the natural laws of flowage, may rise in the wells of such owner and into his pipes, no one of the owners of such lands has the rights, without the consent of all the other owners, to induce an unnatural flow into or through his own wells, or to do any act with reference to the common reservoir, and the body of gas therein, injurious to, or calculated to destroy it. Cited in:

No. 18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.

19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

1900

No. 15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 155 Ind. 545.

"The statute prescribing that it shall be unlawful to conduct natural gas to any point outside of the State is unconstitutional, as affecting interstate commerce, natural gas, when reduced to possession, being an article of commerce, and hence, the Court

will not enjoin such transportation, no undue appropriation, nor the use of artificial means to produce an unnatural flow from the wells being alleged. * * * the distinction between animals *feræ naturæ* and natural gas in respect of their ownership before reduction to possession is a very plain one, and has been clearly pointed out in numerous decisions. * * * In the case of wild animals, before they are reduced to possession, the ownership is in the public, and not in any private person; and they are, therefore, held to be subject to the protection of the sovereign. The privilege of taking, killing, and transporting them may, on this ground, be regulated by the legislature. As to natural gas, however, the public has no title to or control over the gas in the ground. On the contrary, so far as it is susceptible of ownership, it belongs to the owners of the superincumbent lands in common, or, at least, such landowners have a limited and qualified ownership in it to the entire exclusion of the public." Cites:

No. 2.—*State vs. Indiana & Ohio Oil Gas & Mining Co.*, 22 N. E. 778.

4.—*Jamieson vs. Indiana Natural Gas & Oil Co.*, 128 Ind. 555.

Cited in:

No. 19.—*Richmond Natural Gas Co. vs. Enterprise Natural Gas Co.*, 66 N. E. 782.

28.—*Haskell vs. Cowham*, 187 Fed. Rep. 403.

30.—*West vs. Kansas Natural Gas Co.*, 221 U. S. 229

1900

No. 16.—*Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co.*, Supreme Court of Indiana, 155 Ind. 566.

"In a suit to prevent the transportation of natural gas through pipes at a pressure in excess of the natural rock pressure, and by means other than the natural pressure of the gas flowing from the wells, as prohibited by Burns' Rev. St. 1894, Sec. 7507 (Acts 1891, p. 89), a complaint which does not aver that complainants' property is endangered by the defendant's alleged wrongful acts, nor that they are likely to sustain any special injury peculiar to themselves, is demurrable." Cited in:

No. 19.—*Richmond Natural Gas Co. vs. Enterprise Nat. Gas. Co.*, 66 N. E. 782.

1901

No. 17.—*Manufacturer's Gas & Oil Co. vs. Indiana Natural Gas & Oil Co.*, Supreme Court of Indiana, 156 Ind. 679.

This involved the same demurrable law points as the preceding case. Cited in:

No. 19.—*Richmond Nat. Gas Co. vs. Enterprise Natural Gas Co.*, 66 N. E. 782.

1902

No. 18.—*Federal Oil Company vs. Western Oil Company*, U. S. Circuit Court of Appeals, 121 Fed. Rep. 674.

"The nature of property in natural gas and oil contained in the earth, and the legal effect of the instrument here in question, have been settled authoritatively by the rulings of the Supreme Court of Indiana." Cites:

No. 2.—*State vs. Indiana & Ohio Oil, Gas & Mining Co.*, 22 N. E. 778.

5.—*People's Gas Co. vs. Tyner*, 131 Ind. 277.

13.—*Ohio Oil Co. vs. Indiana*, 177 U. S. 190.

14.—*Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co.*, 155 Ind. 461.

1903

No. 19.—*Richmond Natural Gas Co. vs. Enterprise Natural Gas Co.*, Appellate Court of Indiana, 66 N. E. 782.

"The Indiana law providing that natural gas shall not be transported through pipes at a pressure exceeding 300 lb. per square inch, nor otherwise than by the natural

pressure of the gas flowing from the wells, does not prohibit the use of pumps to aid transportation where the pressure is not thereby increased beyond the legal limit, nor does it prohibit the waste of gas; and hence a right in an adjoining owner, tapping a common reservoir, to injunctive relief, is not made out by merely alleging the use of pumps whereby gas is wasted. * * * We cannot agree with counsel for appellees that the legislature, by this statute, has conclusively determined that the effect of the use of pumps is necessarily injurious, and has absolutely prohibited their use. Appellee's case is not made out by merely showing the use of pumps for the purpose of transportation. The use of pumps or compressors for such purpose is not *ipse facto* unlawful. The constitutional validity of the statute is not questioned. * * * It is no longer an open question in this state that natural gas, when reduced to possession, becomes private property, and is a commercial commodity, which the owner may dispose of in whatever manner he may consider most advantageous. * * * Whatever gas, through natural causes, rises to the surface of the ground through the wells of an individual owner, and is carried by such natural forces into tanks, pipe lines, or other receptacles of such owner, becomes absolutely his property. The only limitation upon such owner taking the gas is the manner in which he shall take it from the wells. * * * 'Natural gas in the ground is so far the subject of property rights in the owners of the superincumbent lands that while each of them has the right to bore or mine for it on his own land, and to use such portion of it as, when left to the natural laws of flowage, may rise in the wells of such owner, and into his pipes, no one of the owners of such lands has the right, without the consent of all the other owners, to induce an unnatural flow into or through his own wells, or to do any act with reference to the common reservoir and body of gas therein injurious to or calculated to destroy it.' * * * The sole question here is whether the special findings show that the use of the pumps and compressors will work such injury to appellees, by increasing the flow of gas from the wells, as entitles them to injunctive relief. It is a property of gas, that, while easily compressed, it is always ready to expand and occupy more space. This expansive force or tension imparted to it by some pressure brings it through the wells to the surface, and into pipes or other receptacles. The quantity of the gas that this expansive force or tension, overcoming the atmospheric pressure, will bring to the surface, and into the pipes or receptacles, is the natural flow of the wells. The natural import of the term 'natural flow' as used in the statute, necessarily means the entire volume of gas that will issue from the mouth of a well when retarded only by the atmospheric pressure. If certain wells will bring into a receptacle at the surface 1,000,000 ft. of gas per day, that is their natural flow. Suppose a valve in this receptacle permits one-half this volume to pass into pipes to consumers. Opening an additional valve, and permitting one-half the balance to pass from the receptacle to other consumers, does not increase the natural flow, in the sense in which the term is used in this statute. With the additional valve a greater portion of the natural output of the wells is taken, but the natural flow of the wells has not been increased. And as illustrated in appellant's brief, suppose pumps are attached to a well naturally producing 1,000 bbl. of water per day, for the purpose of forcing 500 bbl. daily to a higher altitude, in order that it may be utilized. Although the water flowed from the well into a main to which the pumps were attached, and the total output put under control, it could not be claimed that the natural flow of the well had been increased by the use of the pumps in carrying half the natural flow to a place for convenient distribution. It must be conceded that the appellant has the right to use all the gas that naturally comes through its wells to the surface of the ground. When we speak of this as the 'natural flow of the wells,' we necessarily mean the total output of the wells from natural causes. The term 'natural flow,' as used in the statute, can be given no other reasonable meaning. * * * Increasing this natural flow of gas or this total output of the wells from natural causes is the thing that is prohibited, and so long as the appli-

ances take less than this natural flow, as thus understood, it cannot be said that they have increased the natural flow. Upon the facts found, all the gas that goes into the pumps on the intake side goes in by reason of its own expansive force or tension. This expansive force or tension must necessarily exist as long as back pressure is maintained at the pumps. * * * Unless the pumps were so operated as to entirely remove this back pressure and create suction in the wells, it could not be said that they would increase the natural flow or natural output of the wells. From the facts found, the only effect of the use of the pumps is to decrease the force of this back pressure, but this the statute does not prohibit. It is true that through the use of the pumps a greater portion of the natural flow or total natural output of the wells will be taken than would be taken without their use, and that the back pressure in the wells will be correspondingly decreased. * * * It cannot be denied that appellant might lawfully dispose of the total product of its wells to consumers near the wells, and the total natural flow be consumed by them, without maintaining any back pressure in the wells. * * * While the court found, as an ultimate fact, that the use of the pumps has the effect of increasing the general flow of gas from the wells, it also found the primary facts from which it appears that the use of the pumps would not increase the amount of gas coming from the wells through the natural laws of flowage. In such case the ultimate fact will be disregarded on appeal." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

4.—Jamieson vs. Indiana Nat. Gas & Oil Co., 128 Ind. 555.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

14.—Manufacturers' Oil & Gas Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 461.

15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

16.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 566.

17.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 156 Ind. 679.

1903

No. 20.—Consumers' Gas Trust Co. vs. American Plate Glass Co., Supreme Court of Indiana, 68 N. E. 1020.

"The mere fact that the complaint disclosed that the gas company was using a pumping station in the winter time, at a point some 6 miles from the land in question, for the purpose of overcoming the friction incident to the flow of gas through many miles of pipe, does not stamp such company as a lawbreaker. There is nothing to suggest that by means of such pumping station the natural pressure of the gas at the wells was increased, or that the pressure in the pipe exceeded 300 lb. per square inch."

1903

No. 21.—Acme Oil & Mining Co. vs. Williams, 74 Pac. Rep. 296.

"Leases are only valuable on development, and are then only valuable to both parties to the extent that the product may be secured and disposed of; and when the only consideration for the lease is the share which the lessor will obtain of what is produced, there is always an implied covenant that diligence will be used toward such production. There are few other mining enterprises where delay is so dangerous, and where diligence in securing immediate possession of the mineral is so necessary, as is mining for oil. As to the precious metals, fixed in the veins which hold them, they remain intact until extracted. Oil, on the contrary, is of a fluctuating, uncertain, fugitive nature, lies at unknown depths, and the quantity, extent and trend of its flow are uncertain. It requires but a small surface area, in what is known as an oil district,

upon which to commence operations for its discovery. But when a well is developed the oil may be tributary to it for a long distance through the strata which holds it. This flow is not inexhaustible, no certain control over it can be exercised, and its actual possession can only be obtained, as against others in the same field, engaged in the same enterprise, by diligent and continuous pumping. It is anybody's property who can acquire the surface right to bore for it, and when the flow is penetrated, he who operates his well most diligently obtains the greatest benefit, and this advantage is increased in proportion as his neighbor similarly situated neglects his opportunity."

1905

No. 22.—*Brewster vs. Lanyon Zinc Co.*, U. S. Circuit Court of Appeals, 140 Fed. Rep. 801.

"Experience in other oil and gas fields has demonstrated that wells drilled in the vicinity of producing wells were not infrequently unproductive. The only method of certainly determining whether or not particular lands contained oil or gas in paying quantity was by drilling thereon to considerable depth. * * * Light will be thrown upon the language used, and the intention of the parties will be better reflected, if consideration is given to the peculiar and distinctive features of the mineral deposits which are the subjects of the lease. Oil and gas are usually found in porous rock at considerable depth under the surface of the earth. Unlike coal, iron, and other minerals, they do not have a fixed situs under a particular portion of the surface, but are capable of flowing from place to place and of being drawn off by wells penetrating their natural reservoir at any point. They are part of the land, and belong to the owner so long as they are in it, or are subject to his control; but when they flow elsewhere, or are brought within the control of another by being drawn off through wells drilled in other land, the title of the former owner is gone. So, also, when one owner of the surface overlying the common reservoir exercises his right to extract them, the supply as to which other owners of the surface must exercise their rights, if at all, is proportionally diminished. * * * By reason of the conditions on which the lease is granted the lessor retains at least a contingent interest in the oil and gas, to the profitable extraction of which the operations are directed. This interest in the subject of the lease, and the fact that the substantial consideration for the grant lies in the provisions for the payment of royalty in kind and in money on the oil and gas extracted, make the extent to which and the diligence with which the operations are prosecuted of immediate concern to the lessor. If they do not proceed with reasonable diligence, and by reason thereof the oil and gas are diminished or exhausted through the operation of wells on adjoining lands the lessor loses, not only royalties to which he would otherwise be entitled, but also his contingent interest in the oil and gas which thus passes into the control of others." Cites:

No. 8.—*Brown vs. Spilman*, 155 U. S. 665.

13.—*Ohio Oil Co. vs. Indiana*, 177 U. S. 190.

1908

No. 23.—*Calor Oil & Gas Co. vs. Franzell et al.*, Court of Appeals of Kentucky, 109 S. W. Rep. 331.

"One who illegitimately wastes or destroys the gas of a district may be punished under the criminal statutes of the State, and may also be enjoined from committing such wrongful acts; but all parties owning gas wells in the district are free to make any legitimate use of the gas they choose, and the fact that this legitimate use tends to exhaust the supply gives the other owners of gas wells in the district no just ground of complaint. The Court knows, as a part of the history of the country, that natural gas districts after flourishing for a while are frequently entirely exhausted, and that manufacturing and power plants established in a district, and dependent upon the use of

the gas for fuel, are forced to move elsewhere. This may happen to the district under consideration; but, as said before, if this results from legitimate sales by the various owners to other customers, no one of them has a just ground for complaint. They have all enjoyed the property while it existed, and when it is exhausted they, of course, can no longer enjoy it."

1908

No. 24.—*Louisville Gas Co. et al. vs. Kentucky Heating Co.*, Court of Appeals of Kentucky, 111 S. W. Rep. 376.

"The right of the surface owners to take gas from subjacent fields or reservoirs is a right in *common*. There is no property in the gas until it is taken. Before it is taken it is fugitive in its nature, and belongs in common to the owners of the surface. The right of the owners to take it is without stint; the one limitation being that it must be taken for a lawful purpose and in a reasonable manner."

1909

No. 25.—*Kansas Natural Gas Co. vs. Haskell*, United States Circuit Court, 172 Fed. Rep. 545.

"From all of which it becomes apparent the contention of defendants that the natural gas found within the territorial limits of the State is the common heritage of the people of the State, which may be conserved and preserved by the State as trustee of those things in which the people have a common interest, as flowing streams, wild animal life, etc., is unsound and must be denied. On the contrary, it must be held he who by lawful right reduces to his possession mineral, gas, or oil, has the same absolute right of property therein, with the same power of barter, sale, or other disposition, including, of necessity, the right of transportation and delivery under such reasonable rules and safeguards as the exigencies of the case may demand and the State employ, as the farmer has of his corn, his wheat, or his stock, or the merchant of his wares, and such absolute right therein as the State cannot deny him without making just compensation, and any attempt to do so would be in violation of the fourteenth amendment to the federal Constitution." Cites:

No. 3.—*Westmoreland, etc., Co. vs. DeWitt*, 130 Pa. 235.

4.—*Jamieson vs. Indiana Natural Gas & Oil Co.*, 128 Ind. 555.

8.—*Brown vs. Spilman*, 155 U. S. 665.

10.—*Thomas C. Kelley vs. Ohio Oil Co.*, 57 Ohio State, 317.

11.—*State of Indiana vs. Ohio Oil Co.*, 150 Ind. 21.

13.—*Ohio Oil Co. vs. Indiana*, 177 U. S. 190.

Cited in:

No. 30.—*West vs. Kansas Nat. Gas Co.*, 221 U. S. 229.

1910

No. 26.—*Ilo Oil Co. vs. Indiana Natural Gas & Oil Co.*, 174 Ind. 635. Indiana Supreme Court.

"A plaintiff that is pumping oil by the use of artificial means has no standing in court to enjoin defendant from doing the same thing, but in a greater degree, since the plaintiff must do equity and come into court with clean hands."

1910

No. 27.—*Kolachny vs. Galbreath*, 26 Oklahoma 772.

"Oil and gas, while in the earth, unlike solid minerals, are not the subject of ownership distinct from the soil, and the grant of the oil and gas, therefore, is a grant, not of the oil that is in the ground, but of such part as the grantee may find and passes nothing that can be the subject of an ejectment or other real action." Cited in:

No. 29.—*Frank Oil Co. vs. Bellview Gas & Oil Co.*, 29 Okla. 719.

1911

No. 28.—Haskell vs. Cowham, United States Circuit Court, 187 Fed. Rep. 403.

"The right of a private citizen by means of his ownership of, or of his mining leases of, land to draw gas or oil from beneath its surface is property and sometimes valuable property. * * * Interstate commerce in natural gas, including the transportation thereof in pipe lines, is a subject national in its character and susceptible of regulation by uniform rules, and the silence or inaction of Congress regarding it is a conclusive indication that it intends that interstate commerce therein shall be free and laws of States, and acts of its officers which either prohibit or substantially burden it are without justification violative of the Constitution and void. * * * Neither a State nor its officers may prevent or unreasonably burden interstate commerce in the natural gas within it by preventing the use to transport it of pipe lines across the highways of the State by means of the exercise, or the refusal to exercise, the police powers or the proprietary powers of the State over them." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

15.—Manufacturers' Gas & Oil Co., vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

1911

No. 29.—Frank Oil Co. vs. Bellview Gas & Oil Co., 29 Okla. 719.

"Oil and gas, while in the earth, unlike solid minerals, are not the subject of ownership distinct from the soil, and the grant of the oil and gas, therefore, is a grant not of the oil that is in the ground, but of such part as the grantee may find, and passes nothing except the right to explore for the same under the terms of such contract. * * * A different rule of construction obtains as to oil and gas leases from that applied to ordinary leases or to other mining leases. Owing to the peculiar nature of the mineral, and the danger of loss to the owner from drainage by surrounding wells, such leases are construed most strongly against the lessee and in favor of the lessor. Cites:

No. 27.—Kolachny vs. Galbreath *et al.*, 29 Okla. 772.

1911

No. 30.—West vs. Kansas Natural Gas Co., United States Supreme Court, 211 U. S. 229.

"The case is a valuable one and clearly announces the rights of an owner to the soil beneath it, and the relation of his rights to all other owners of the surface of the soil. The right of taking the gas, it was said, was common to all owners of the surface, and because of such a common right in all land owners, and unlimited use (against a wasteful use the statute was directed) by any it was competent for the State to prohibit. This limitation upon the surface owners of property was justified by the peculiar character of gas and oil, they having the power of self-transmission, and that therefore to preserve an equal right in all surface owners there could not be an unlimited right in any. Gas and oil were likened to, not made identical with, animals *feræ naturæ*, and, like such animals, were subject to appropriation by the owners of the soil, but also, like them, did not become property until reduced to actual possession. But an important distinction was pointed out. In things *feræ naturæ*, it was observed, all were endowed with the power of reducing them to possession and exclusive possession. In the case of natural gas, only the surface proprietors had such power, and the distinction, it was said, marked the difference in the extent of the State's control. In the one, as the public are the owners, everyone may be absolutely prevented from seeking to reduce to possession. No divesting of private property, under such a condition, can be conceived, because the public are the owners, and the enactment by the State

of a law as to the public ownership is but the discharge of the governmental trust resting in the State as to property of that character. * * * On the other hand, as to gas and oil, the surface proprietors within the gas field all have the right to reduce to possession the gas and oil beneath. They could not be absolutely deprived of this right which belongs to them without a taking of private property.' And this right, it was further said, was coequal in all of the owners of the surface, and that the power of the State could be exerted 'for the purpose of protecting all the collective owners, by securing a just distribution, to arise from the enjoyment by them of their privilege to reduce to possession, and to reach the like end by preventing waste.' And further characterizing the statute, it was said, viewed as one to prevent the waste of the common property of 'the surface owners, it protected their property, not divested them of it. And special emphasis was given to this conclusion by the comment that, to assert that the right of the surface owners to take was, under the 14th Amendment, a right to waste, was to say that one common owner may divest all the others of their rights without wrongdoing; but the lawmaking power cannot protect all the owners in their enjoyment without violating the Constitution of the United States.' The case, therefore, is an authority against, not in support of, the contention of the appellant in the case at bar. The statute of Indiana was directed against waste of the gas, and was sustained because it protected the use of all the surface owners against the waste of any. The statute was one of true conservation, securing the rights of property, not impairing them. Its purpose was to secure to the common owners of the gas a proportionate acquisition of it—a reduction to possession and property—not to take away any right of use or disposition after it had thus become property. It was sustained because such was its purpose. * * * Gas, when reduced to possession, is a commodity; it belongs to the owner of the land; and, when reduced to possession, is individual property, subject to sale by him, and may be a subject of intrastate commerce and interstate commerce. * * * These propositions were announced: (1) Natural gas is as much a commodity as iron ore, coal, or petroleum, or other products of the earth, and can be transported, bought and sold as other products. (2) It is not a commercial product when it is in the earth, but becomes so when brought to the surface and placed in pipes for transportation." Cites:

No. 2.—*State vs. Indiana & Ohio Oil, Gas & Mining Co.*, 22 N.E. 778.

No. 4.—*Jamieson vs. Indiana Natural Gas & Oil Co.*, 128 Ind. 555.

No. 13.—*Ohio Oil Co. vs. Indiana*, 177 U. S. 190.

No. 15.—*Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co.*, 155 Ind. 545.

No. 25.—*Kansas Natural Gas Co. vs. Haskell*, 172 Fed. Rep. 545.

Cited in:

No. 31.—*Haskell vs. Kansas Natural Gas Co.*, 224 U. S. 217.

1912

No. 31.—*Haskell vs. Kansas Natural Gas Co.*, United States Supreme Court, 224 U. S. 217.

"Natural gas after severance is a commodity which might be dealt in like other products of the earth, as coal and other minerals, and is a legitimate subject of interstate commerce; and that no State, by such laws as were involved in the case, can prohibit its transportation in interstate commerce beyond the lines of that State. The court held, after considering and construing the provisions of the act of 1907, that it was, upon its face, a law undertaking to prohibit the transmission or transportation in interstate commerce of natural gas to points beyond the State; that it was an unconstitutional interference with the rights of the complainants, who were legitimately engaged in that commerce, and that therefore the act was null and void." Cites:

No. 26.—*West vs. Kansas Natural Gas Co.*, 221 U. S. 229.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Necessary Use and Effect of Gas Compressors on Natural Gas Field Operating Conditions*

BY SAMUEL S. WYER,† M. E., COLUMBUS, OHIO

(New York Meeting, February, 1916)

1. THE following is an abridgment of a recent report made by the author, covering an investigation of:

(A) The necessary use of natural-gas compressors;

(B) The effect of gas compressors on natural-gas field operating conditions in Ohio, with special reference to the working pressure relations of gas wells of one company discharging by natural rock pressure flow without compressors, and adjacent wells of other companies, discharging into intake lines to gas-compressing stations.

Tests were made at 107 different places in the counties of Fairfield, Licking and Knox, Ohio, which required 7,000 miles of automobile driving, and consumed 6 months' time.

2. A proper conception of the necessary use and effect of natural-gas compressors in natural-gas production necessitates a clear understanding of certain fundamental definitions and judicial doctrines, which are given herein.

3. Gas is a fluid composed of a large number of molecules which are vehicles of energy continually in motion, and having an inherent tendency to get farther and farther apart. The range of motion of the molecules is limited only by the volume of the closed containing vessel in which they constantly move to and fro. The most distinguishing characteristic of gas is its universal property of completely filling an inclosed space.

4. Gas pressure is the result of the combined efforts of all the moving molecules in the mass trying to get farther and farther apart, that is, a mass of gas inclosed in a vessel expands and fills it, and being restrained from further expansion, it exercises a pressure against the walls of the vessel. This pressure is the same in all directions on equal areas of surface, as shown in Fig. 1. With a given mass of gas, any increase in volume of containing vessel will give the molecules more range of motion,

* The public's interest in the age and magnitude of the art of gas compressing and development of the law relating to the use of gas compressors in natural gas production is discussed in the preceding paper in this *Bulletin*.

† Consulting Engineer.

and thereby lower the pressure. Thus, if a part of a given mass is removed from a closed vessel or reservoir, the remaining mass of gas will expand instantly and keep the vessel or reservoir *filled*, but at a *lower* pressure.

5. The pressure per unit of area exercised inward upon a mass of fluid is transmitted undiminished in all directions, and acts with the same force upon all surfaces in a direction at right angles to those surfaces. Hence, the pressure applied to any area of a confined fluid is transmitted to every other equal area through all the fluid, to the walls of the containing chamber, without diminution. This is known as Pascal's law, and is shown in Fig. 1. According to this, the gas pressures in the various parts of a

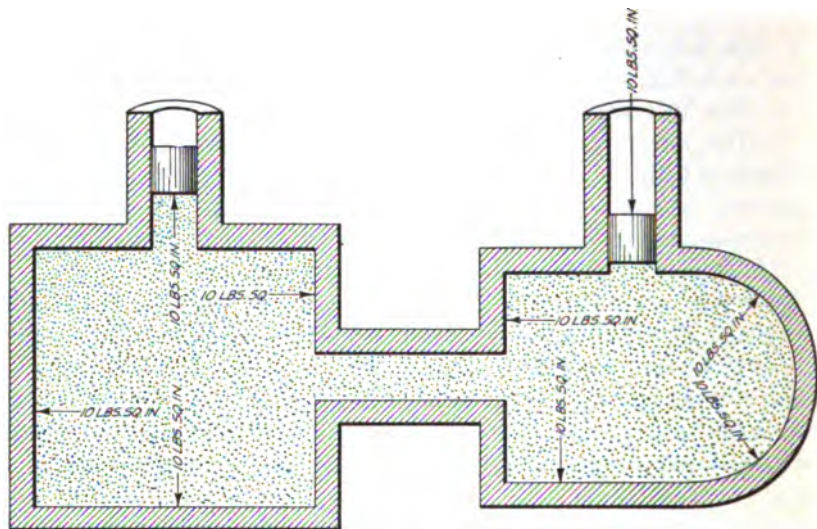


FIG. 1.—DIAGRAM ILLUSTRATING PASCAL'S LAW.

“continuous and connected reservoir” must be equal. If they are not equal, the various parts of the reservoir are not continuous or connected.

6. As long as energy of any form undergoes no transformation it tends to gravitate to a lower degree of intensity, that is, become more stable and approach a universal level of stable equilibrium. Without external assistance fluids can flow only from higher to lower levels. Thus, water always seeks the lowest level, and confined gas tends to expand to lower pressures, and gases confined in “connected or continuous reservoirs” always tend to and must ultimately equalize their pressures, and gas flow in pipes or underground reservoirs is always in the direction of the point of lower pressure.

7. Boyle's law is “The volume of gas is inversely proportional to the absolute pressure to which the gas is subjected; or, what is the same thing, the product of the absolute pressure and the volume of a given quantity

of gas remains constant." This law is not followed literally by natural gas, but various gases deviate from it slightly, varying with the composition of the gas itself, and the pressure to which the gas is subjected.¹

8. The word "barometer" means instrument for measuring weight, and it indicates merely the pressure of the air at the point where it is placed. Sea level is the datum from which atmospheric pressures are reckoned. At that point dry air at 32°F. exercises a pressure of 14.7 lb. per square inch. The relation of atmospheric pressure to the other pressure terms is shown in Fig. 2.

9. The term "vacuum" has three distinct meanings.

(A) Literally, a space entirely devoid of matter. This is merely a theoretical conception.

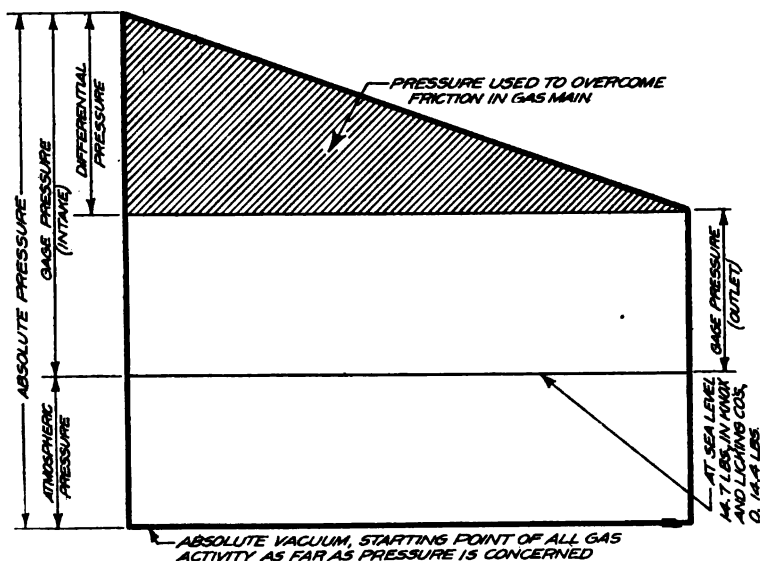


FIG. 2.—DIAGRAM ILLUSTRATING GAS PRESSURE TERMS.

(B) A space nearly devoid of matter, or a vessel from which the air has been exhausted to a very high degree, as may be done by apparatus in a scientific laboratory. Such high exhaustions are called by courtesy, vacua, as they are the nearest approaches that scientists have been able to make to absolute vacuum by the most refined methods known to science, and yet they are far from an absolute vacuum.

(C) Ordinary use of the term means merely a partial diminution of pressure below the normal atmospheric pressure. This is the engineering conception, and the maximum attainable degree with ordinary engineer-

¹ This deviation is discussed in detail in a paper to be presented at the Spring Meeting (April, 1916) of the American Society of Mechanical Engineers.

ing appliances is about 14 lb. below atmosphere. This is the sense in which the term is used herein.

10. The various gas pressure terms are illustrated in Fig. 2.

Gage pressure is simply the pressure indicated by a pressure gage.

Barometric pressure is the atmospheric pressure, measured by a barometer.

Absolute pressure is the sum of the gage pressure and the atmospheric pressure.

Differential pressure is simply the difference between the pressure at the inlet and outlet of a gas line. In gas transmission it is necessary to have a high differential pressure in order to secure enough driving power to force the gas through the main.

11. In measuring vacuum it is important to note that this is always reckoned downward from atmospheric pressure toward absolute vacuum; that is, it is measured in the direction opposite that which ordinary gage pressure or absolute pressure is measured, as shown in Fig. 11.

12. The movements of gases in pipes are controlled by definite laws. The three that relate directly to the gas-compressing problem are:

(A) "Increasing the length of a pipe four times reduces the quantity of gas discharged by the pipe one-half."

(B) "To double the quantity of gas delivered by a pipe it is necessary to quadruple the differential pressure."

(C) "The differential pressure required to pass a given quantity of gas varies exactly as the length of the pipe."

The primary function of gas compressors is to increase the available differential pressure in the line so as thereby to increase the line's carrying capacity.

13. "The term 'natural flow' necessarily means the entire volume of gas that will issue from the mouth of a gas well when retarded only by the atmospheric pressure" (Appellate Court of Indiana. 66 N. E. 782. Richmond Natural Gas Co. vs. Enterprise Natural Gas Co.).

The courts have used the term "natural flow" synonymously for the engineering term "open flow," both, however, meaning exactly the same thing. The marked difference between the open flow of a gas well and the actual flow that may be obtained under routine operating conditions is emphasized in Fig. 9.

14. Rock pressure and volume must decline as gas is removed. When nature generated or deposited the natural gas in the rock reservoir a fixed amount of gas was placed in a fixed inclosing space. The pressure in the rock—called "rock pressure"—was the result of the pressing into this fixed rock space of a larger volume of gas than the mere free air capacity of this rock reservoir. The degree of compression employed by nature in the formation process determined the intensity of the resulting pressure in the reservoir; that is, a high degree of compression produced a high

rock pressure, and a low degree of compression produced a low rock pressure.

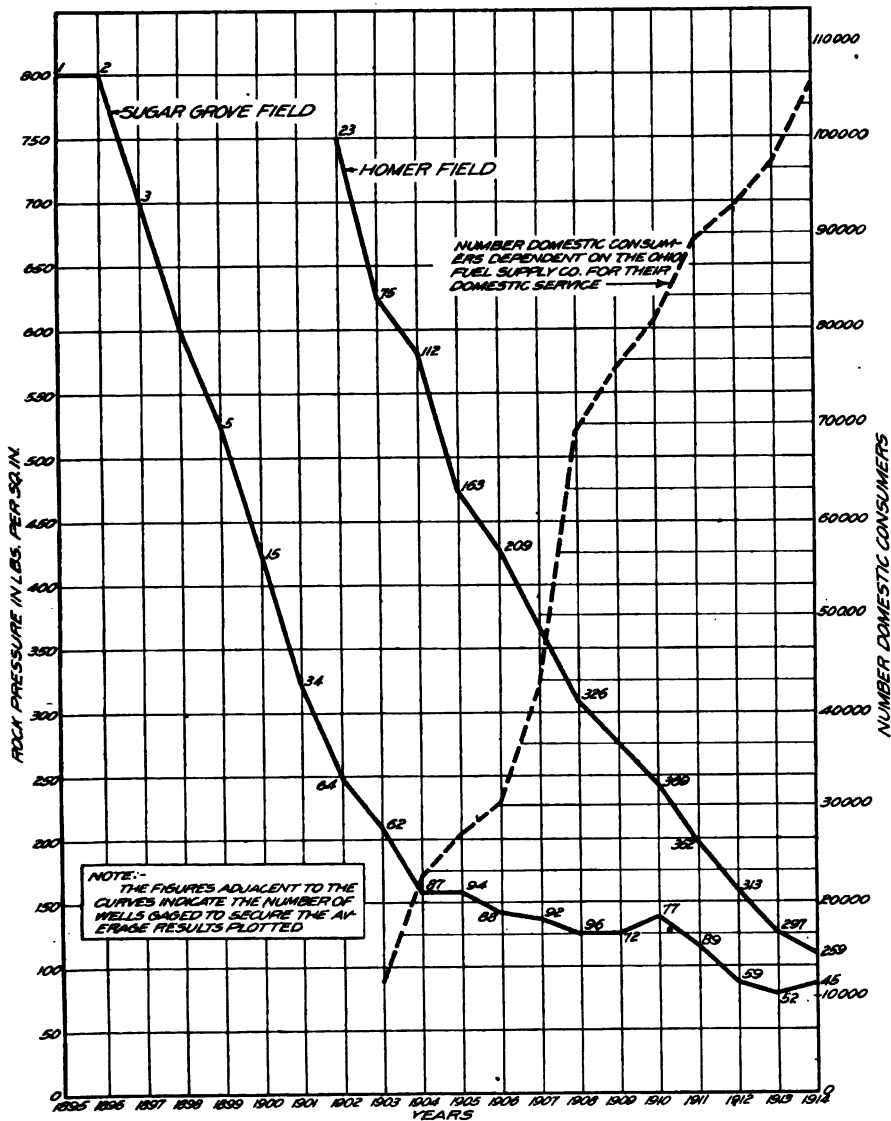


FIG. 3.—GRAPHICAL ILLUSTRATION OF NATURAL DECLINE OF GAS ROCK PRESSURE DUE TO REMOVAL OF GAS IN TWO TYPICAL FIELDS IN OHIO, AND SIMULTANEOUS GROWTH OF NUMBER OF DOMESTIC CONSUMERS.

15. Coming now to the removal of this natural deposit of gas, we are confronted with the following:

1. A fixed volume of the reservoir.

2. A fixed amount of gas inclosed in this fixed reservoir.

3. A certain rock pressure resulting from the contraction of the gas volume into the fixed reservoir.

Now, if a part of this fixed volume of gas is removed by tapping the reservoir from the surface of the earth, the remaining gas expands and keeps the reservoir completely filled, in accordance with the principles stated in Sections 4 and 5. The change in pressure will be in accordance with the law stated in Section 7. From this it is evident that, regardless of how the gas may be removed—whether by natural flow or by means of compressors—a decline in the rock pressure and total volume is inevitable. The decline in rock pressure in two typical fields in Ohio is shown in graphical form in Fig. 3.

16. Decrease in rock pressure always lowers compressor capacity. The rock pressure of wells has a direct bearing on the intake pressure that may be maintained at the gas compressors. The output of a typical compressor operating against a discharge pressure of 300 lb. gage is as follows, for the respective intake pressures:

Intake Pressure above Atmosphere	Capacity in Cubic Feet Free Gas per 24 Hr. Based on 14.4 Lb. Atmos- pheric Pressure
150	30,800,000
100	20,700,000
75	15,600,000
50	10,500,000
30	6,550,000
20	4,170,000

Thus the decline of the wells reduces the capacity of the compressing station and necessitates the installation of more compressors as the field grows older.

17. Judicial recognition of declining gas volume:

United States Supreme Court:

"Oil and gas have no fixed situs under a particular portion of the earth's surface within the area where they abound. They have the power, as it were, of self-transmission. No owner of the surface of the earth within the area beneath which oil and gas move can exercise his right to extract from the common reservoir in which the supply is held without to an extent diminishing the source of supply as to which the other owners of the surface must exercise their rights."

"If possession of the land is not necessarily possession of the oil and gas, is there any reason why an oil and gas operator should not be permitted to adopt any and all appliances known to the trade to make the production of his wells as large as possible."—Ohio Oil Co. vs. Indiana. 177 U. S. 190.

United States Circuit Court:

"Oil and gas, unlike coal and iron, and other minerals, do not have a fixed situs under a particular portion of the surface, but are capable of flowing from place to place and of being drawn off by wells entering their natural reservoir at any point. They are part of the land and belong to the owner so long as they are in it or subject

to his control, but when they flow elsewhere and are brought within the control of another by being drawn off by wells drilled in other land, the title of the former owner is gone. So, also, when one owner of the surface overlying the common reservoir exercises his right to extract them the supply as to which other owners of the surface must exercise their right if at all is proportionately diminished."—*Brewster vs. Lanyon Zinc Co.* 140 Fed. Rep. 801.

Supreme Court of Pennsylvania:

"The mere fact that gas-mining operations upon one tract of land are taking gas from the earth and thereby diminishing the quantity of gas which otherwise would go to wells on adjacent tracts of land furnishes no ground for complaint for culpable interference."—*Hague vs. Wheeler.* 157 Pa. 324.

18. Relation of known and unknown features of gas compression question:

Known

1. Gas is a mineral with a fugitive and wandering existence, and has a tendency to escape without the volition of the owner.
2. Gas flow follows definite physical laws.
3. There is no regeneration, as shown by the large number of abandoned wells.
4. There are reservoirs of fixed volume, each containing a fixed amount of gas.
5. The removal of a part of the volume of gas in the reservoir must reduce the reservoir pressure.

Unknown

1. Gas cannot be located from the surface, as so well stated by the Indiana Supreme Court:

"We judiciously know, as a matter of common knowledge, that gas or oil does not exist in paying quantities under all the lands within the recognized district, and there is no other generally acknowledged way than putting down a well to determine whether or not it does exist."—162 Ind. p. 326. *Consumers' Gas Trust Co. vs. Littler.*

2. The existence of the underground reservoirs cannot be determined from the surface.

3. There is no way of determining the direction or magnitude of underground gas flow. There is absolutely no way of determining how much of the gas removed comes from under the leases owned and how much may be supplied by adjacent leases, or what the limits of the underground reservoirs supplying the various wells may be.

4. There is no way of definitely determining the origin of natural gas.

19. Judicial dicta relating to possession of gas:

United States Supreme Court:

"Petroleum gas and oil are substances of a peculiar character. They belong to the owner of the land and are part of it so long as they are on it or in it, or subject

to his control, but when they escape and go into other land, or come under another's control, the title of the former owner is gone. If an adjoining owner drills his own land and taps a deposit of oil or gas extending under his neighbor's field so that it comes into his well it becomes his property."—*Brown vs. Spilman*. 155 U. S. 665.

"The property of the owner of lands in oil and gas is not absolute until it is actually in his grasp and brought to the surface."—*Ohio Oil Co. vs. Indiana*. 117 U. S. 190, affirming Indiana Supreme Court.

Pennsylvania Supreme Court:

"Plaintiff assumes that there is a certain fixed amount of oil and gas under his farm in which he has an absolute property. True they belong to him while they are a part of his land, but when they migrate to the land of his neighbor, or become under his control, they belong to the neighbor."—*Jones vs. Forest Oil Co.* 194 Pa. 379.

"Possession of the land is not necessarily possession of the gas. If an adjoining or even a distant owner drills his own land and taps your gas so that it comes into his well and under his control, it is no longer yours, but his. The one who controls the gas—has it in his grasp, so to speak—is the one who has possession in the legal as well as ordinary sense of the word."—*Westmoreland, etc., Co. vs. DeWitt*. 130 Pa. 235.

20. The engineering features relating to possession of gas are shown in Fig. 4. By means of the rubber, the packer is firmly anchored into the rock, and in this way compels the gas to go up to the surface on the inside of the tubing. Wells are ordinarily about half a mile deep, and as 2-in. tubing is usually used, it is evident that considerable friction must be overcome from the bottom of the well to the surface. In general, for this reason, several pounds of vacuum at the surface would require more than atmospheric pressure at the packer to bring the gas from the packer to the surface of the ground. Hence, even a marked vacuum at the mouth of the well does not necessarily mean less than atmospheric pressure at the gas sand. From an engineering point of view, as soon as the gas enters the gas well packer it comes "into the grasp" or "under the control" of that well owner.

21. Gas is compressed merely to expedite transmission—for the same reason that makes it necessary to compress cotton, hay, or straw, for shipment. The first feature is to contract the volume, and secondly, to secure large enough differential pressure drop between the intake and discharge of the transmission line to force the gas through the line. The effect of gas compression on its volume is shown graphically in Fig. 5.

22. The practical necessity for using gas compressors is evident from the graphical data shown in Fig. 6. It will be noted here that the average intake pressure at Homer is about 1 lb., and that this must be increased to 200 lb. by the compressors in order to transmit gas a distance of 137 miles to Cincinnati, Ohio. Furthermore, the terminal pressure at the Cincinnati end of the line is much larger than the intake pressure at Homer. It is necessary to maintain this terminal pressure in order to get the gas through the medium- and low-pressure lines to the Cincinnati

distributing system and service lines to consumers' house piping. The large variation in the consumers' demands on this transmission line is shown by the volume curve, and ability to take care of this large variation in volume is necessary in order that adequate service may be rendered

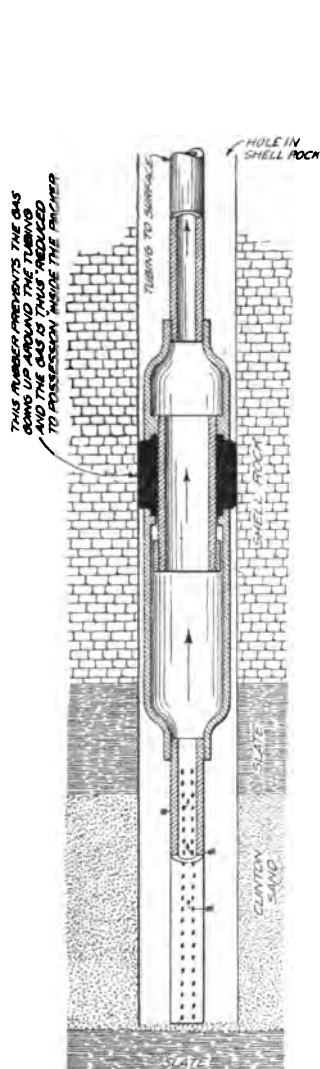
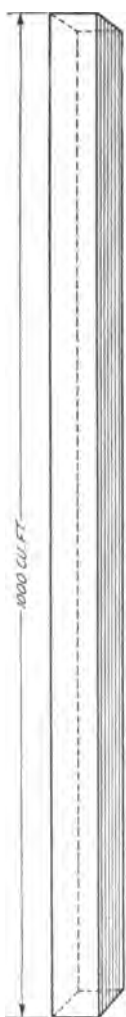


FIG. 4.—PACKER AT BOTTOM OF GAS WELL.



GAGE PRESSURES—4 OZ. ($\frac{1}{2}$ LB.) PER SQ. IN. — 300 LBS. PER SQ. IN.
HEAT UNITS — 1,000,000 — 1,000,000
VOLUME — 1000 CU. FT. — 46 CU. FT.

FIG. 5.—EFFECT OF PRESSURE ON GAS VOLUME.

to the consumer. This shows at once how the entire production end of the natural-gas business must be subordinated to the consumers' demands.

23. The terms "gas compressing" and "gas pumping"—unfortu-

nately—are almost universally used synonymously to describe the contraction of volume of gas by compressing it with a machine known as a gas compressor.

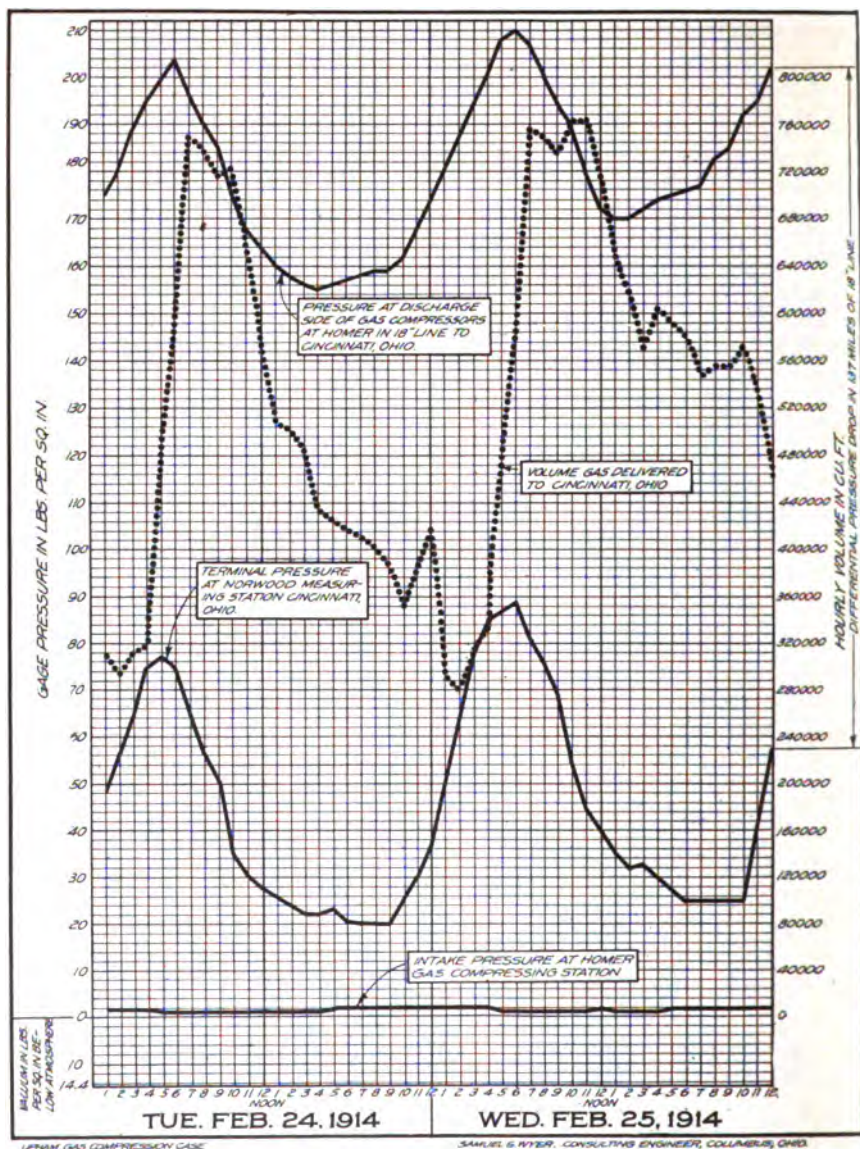


FIG. 6.—TYPICAL OPERATING CONDITIONS ON OHIO FUEL SUPPLY CO.'S 18-IN MAIN LINE TO CINCINNATI, OHIO.

24. Contrary to a widespread popular notion, the compression of natural gas does not decrease its heating value. While a certain amount

of gas is used to drive the compressors, this does not in any way affect the heating value of the gas passing through the compressors. The only shrinkage will be gasoline that condenses in the transmission lines, and the percentage thus removed is usually so small, considering the large volume of gas handled, as to be negligible. In a recent test this was found to be less than $\frac{1}{6}$ of 1 per cent. In view of the preceding facts, the heating value per cubic foot of gas at the end of the transmission lines will be substantially the same as in the gas field.

25. Features of natural-gas business making gas compressors necessary:

(A) The natural-gas business is unique in that it combines a hazardous mining operation with a public utility service.

(B) Natural gas, though a mineral and obtained by mining operations, with more uncontrollable and uncertain features to cope with than exist in the mining of coal or other minerals, when served to the public becomes a utility service and requires constant attention from the time it is reduced to possession at the bottom of the well, and embodies an unbroken chain of service features until it is burned at the consumers' fixtures.

(C) The consumer's interest and rights extend clear back to and depend on the gas wells and reserve acreage that a producing company maintains to protect such wells and insure an adequate present and continuous future service.

(D) Natural-gas service is instantaneous. There can be no delays in rendering service, as is possible—and universally practised—in other transmission agencies, such as railroad and traction lines. For instance, a railroad train can very easily start an hour later, in case of congestion of traffic, but natural-gas service that delivers gas for cooking breakfast an hour after the consumer needs it would not only be useless to the consumer, but would not be tolerated in any community.

(E) The market for natural gas is not fixed, but varies largely with the seasons of the year, time of day, temperature of atmosphere, locality, and consumer's caprice in using or not using gas on his premises.

(F) The gas consumers will not contract for or agree to use a fixed amount of gas each day, but are entitled by law to take gas as they need it.

(G) Storage facilities are not commercially feasible for storing natural gas at the intake end, in transit, or at the delivery end of transmission line.

(H) Gas must be sold at the delivery end at a price fixed by law for a period of years.

(I) When the fixed price is fair and reasonable—that is, when its operation will yield a net return to the gas company commensurate with the value of the service it is rendering and the hazards of the business it is engaged in—it becomes the duty of the gas company:

First, to conserve the supply of gas in every way possible. By conservation is meant not merely saving, but using in the most effective manner. This means that it is the duty of the gas company—under such conditions—to remove every foot of gas from the ground that can be obtained.

Second, every appliance known to the art ought to be used to bring about the most economical mining of the gas and most effective method of transmission and distribution.

(J) A normal characteristic of every gas field is that its rock pressure decreases each year as the gas is being removed from the ground, as shown in graphical form in Fig. 3.

(K) In the face of a marked decline in rock pressure, the number of domestic consumers is on the increase (see Fig. 3), thus augmenting the difficulties under which the natural-gas company must operate in order to render service to its consumers. This means that as the fields grow older it is necessary for the gas company to supplement the rapidly declining pressure by artificial means.

26. In the tests herein described two general classes of data were obtained, as discussed in the following two sections:

27. Simultaneous recording pressure-gage records were made, showing actual routine operating conditions in various parts of the field; 17 standard recording pressure gages with 8-in. charts were used for this service; 290 recording pressure-gage charts, made at 43 different places in the field, were obtained during the months of August, September, October, November and December, 1913.

28. The relation of the natural or open flow to actual line flow from various wells was obtained by the use of ordinary Pitot tubes. The open flow was determined by the Pitot tube arrangement shown in Fig. 7. A short length of pipe was screwed into the top of the well gate, and as the well discharged into the atmosphere the dynamic pressure corresponding to the velocity of the gas was measured in the U-tube of the pressure gage shown at the right. The actual line flow was determined by the Pitot tube arrangement shown in Fig. 8. This was placed in the discharge line of the well when the well was working under routine conditions. The static pressure was observed by the spring pressure gage at the top and the dynamic differential pressure was obtained by the U-tube fluid gage as shown. A square-jawed micrometer caliper was used for measuring the difference in liquid elevations in the U-tubes. The volume passing both Pitot tubes was computed by the ordinary Pitot tube tables. A summary of some of the tests is given in Fig. 9.

29. The precautions taken to secure accuracy were as follows: All the recording gages used were new, and these and all other gages used in this series of tests were carefully calibrated before being placed into service and during service by testing on a standard gage-testing apparatus.

A dependable watch, running on central standard time, was used as the basis for starting the clocks of the various recording gages. This watch was carefully checked at least three times a week against the standard time furnished by the Western Union Telegraph Company, and every time a chart was set on a gage the clock was carefully set to correspond with the correct time by the watch. In this way it was an easy matter to attain synchronous action from all the recording gages. Each recording

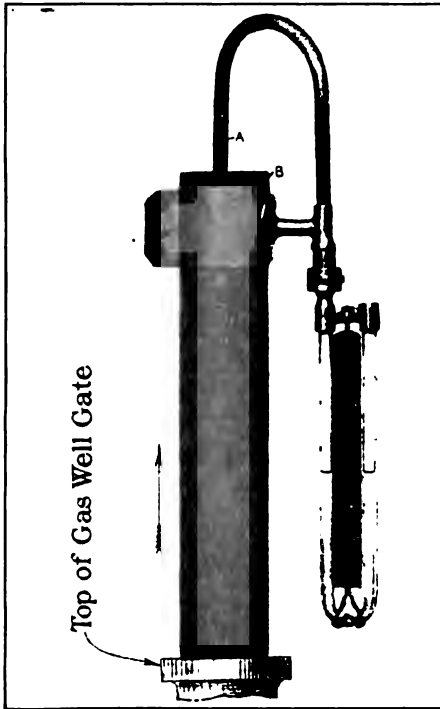


FIG. 7.—PITOT TUBE TO DETERMINE OPEN FLOW OF GAS WELL.

The end of the Pitot tube *A* and the end of the pipe *B* are in the same horizontal plane. The end of tube *A* is beveled so that its inside circumference will present a sharp annular edge against the flow of gas.

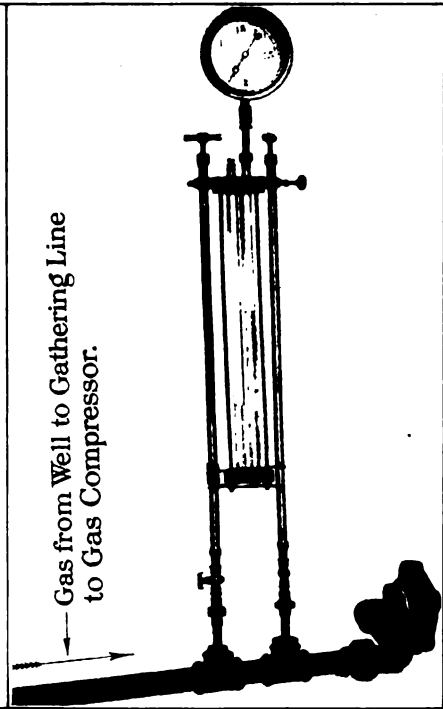


FIG. 8.—PITOT TUBE FOR DETERMINING ACTUAL LINE FLOW FROM GAS WELL.

gage, when in service, was also locked and sealed, to prevent tampering. Every chart in this entire series of tests was set and removed from the gage by the author.

30. The data obtained by the recording gage charts were plotted on 116 sheets of uniform size and scale, of which two are given in Figs. 10 and 11. Each sheet gives the results of 48 consecutive hours of service. The gage-pressure scale is given on the left-hand side of the sheet and the

NAME OF WELL	FRANK WHITE	WILMOT SPERRY	ELLEN M. JACKSON	W. L. PARROT	G. R. GEORGE	D. W. WALLACE	ELIZABETH HAUSER	CHARLES CAREY	W. C. COOPER	W. O. HAMMINS
TOWNSHIP	COLLEGE	CLINTON	MILFORD	CLINTON	CLINTON	MORGAN	CLINTON	LIBERTY	CLINTON	CLINTON
DATE	NOV. 26	NOV. 21	NOV. 27	DEC. 22	NOV. 20	DEC. 22	NOV. 22	NOV. 26	DEC. 21	DEC. 21
GAGE PRESSURE AT WELL	35 LBS.	17 LBS.	22 LBS.	14 LBS.	20 LBS.	10 LBS.	17 LBS.	15 LBS.	16 LBS.	16 LBS.
GAGE PRESSURE AT COMPRESSOR INTAKE	8 LBS.	4 LBS.	14 LBS.	1 LBS.	5 LBS.	1 LBS.	4 LBS.	8 LBS.	5 LBS.	5 LBS.
PRESSURE DROP BETWEEN WELL AND COMPRESSOR	27 LBS.	13 LBS.	8 LBS.	13 LBS.	15 LBS.	9 LBS.	13 LBS.	7 LBS.	11 LBS.	11 LBS.
NATURAL OR OPEN FLOW IN 24 HOURS CU. FT.	49500	45000	132000	35000	70000	74000	50000	49600	100000	87000
ACTUAL FLOW INTO LINE IN 24 HOURS CU. FT.	16000	24000	76800	45600	43000	48000	33600	33600	86000	76900
PERCENT OF OPEN NATURAL FLOW USED	32	53	58	60	61	64	67	68	86	88

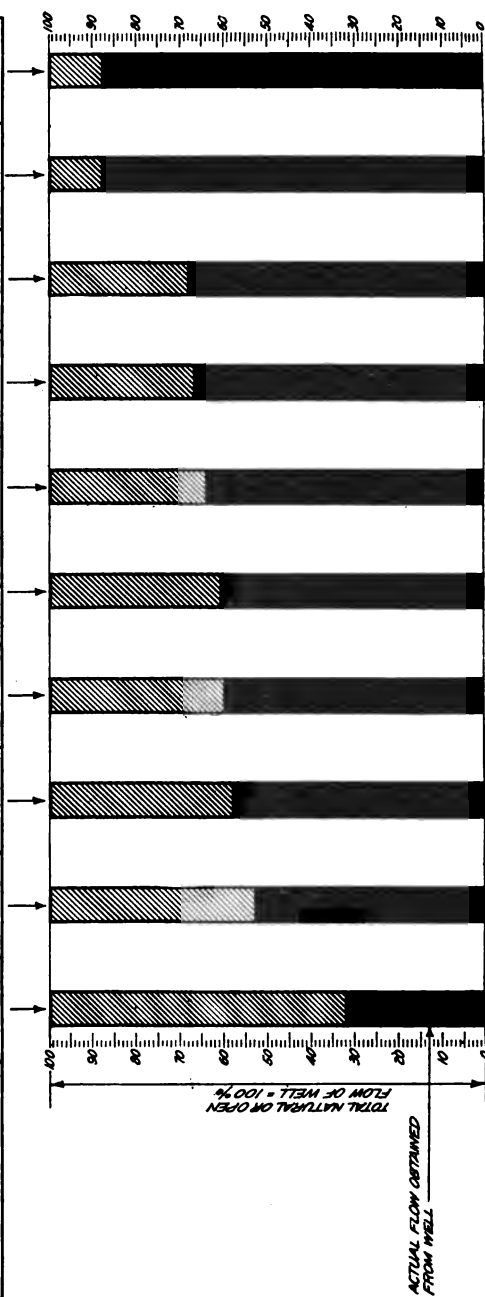


FIG. 9.—NATURAL OR OPEN FLOW, AND ACTUAL LINE FLOW FROM VARIOUS WELLS OF OHIO FUEL SUPPLY CO. IN KNOX COUNTY, OHIO, IN 1913, DISCHARGING INTO INTAKE LINES TO HOMER GAS-COMPRESSING STATION.

absolute-pressure scale on the right-hand side, while the dates are always marked at the bottom.

31. The observations and data obtained led to the following conclusions:

(A) Ordinarily only a relatively small percentage of the natural flow may be obtained from a natural-gas well under routine operating condi-

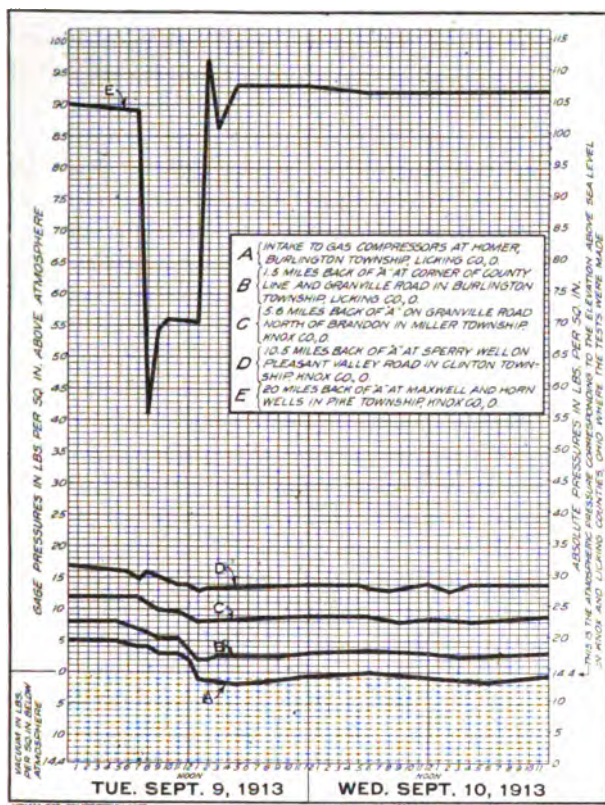


FIG. 10.—SHORT RANGE OF LOW PRESSURES IN "MILLER INTAKE LINE" OF OHIO FUEL SUPPLY CO., DISCHARGING IN HOMER COMPRESSING STATION, BASED ON SIMULTANEOUS RECORDING PRESSURE-GAGE RECORDS.

tions, even when such a well is discharged into the intake line to a gas-compressing station.

(B) That this Ohio field is not made up of one large "connected and continuous reservoir," but rather composed of many separate and discontinuous reservoirs, is evident from:

(a) The large number of dry holes drilled.

(b) The large number of wells that have been abandoned in close proximity to producing wells.

- (c) The continuous difference in rock pressure in adjacent wells that have been shut in for a period of years.
- (d) The marked variation in working pressures of adjacent wells.
- (e) The marked variation in rock pressure in different parts of the field.
- (C) The diminution in rock pressure has simply been the normal

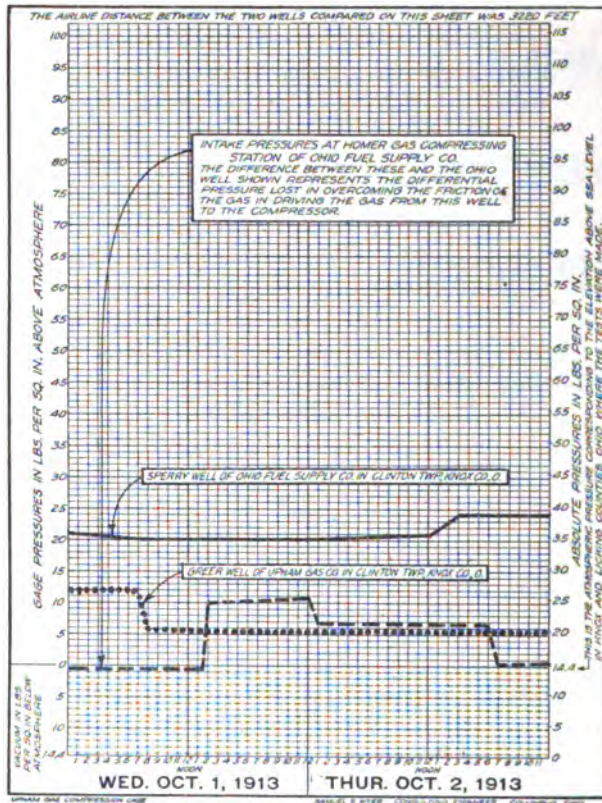


FIG. 11.—WORKING PRESSURE RELATIONS OF GAS WELL OF UPHAM GAS CO., DISCHARGING INTO LINE BY NATURAL ROCK PRESSURE WITHOUT COMPRESSORS, AND ADJACENT GAS WELL OF OHIO FUEL SUPPLY CO., DISCHARGING INTO INTAKE LINE TO HOMER GAS COMPRESSING STATION BASED ON SIMULTANEOUS RECORDING PRESSURE-GAGE RECORDS.

decrease that is inevitable with the removal of the gas from the various reservoirs.

(D) The water troubles that are experienced with certain wells in the field are due to the decrease in pressure in the local reservoir supplying the particular well, caused by the removal of the gas from such reservoir, regardless as to whether such removal was by letting the wells discharge into lines without compressors, or with compressors.

(E) A comparison of the area of territory held to maintain and protect each producing well shows that the large companies using gas compressors maintain about 10 times more protection for their wells than the small companies not using gas compressors.

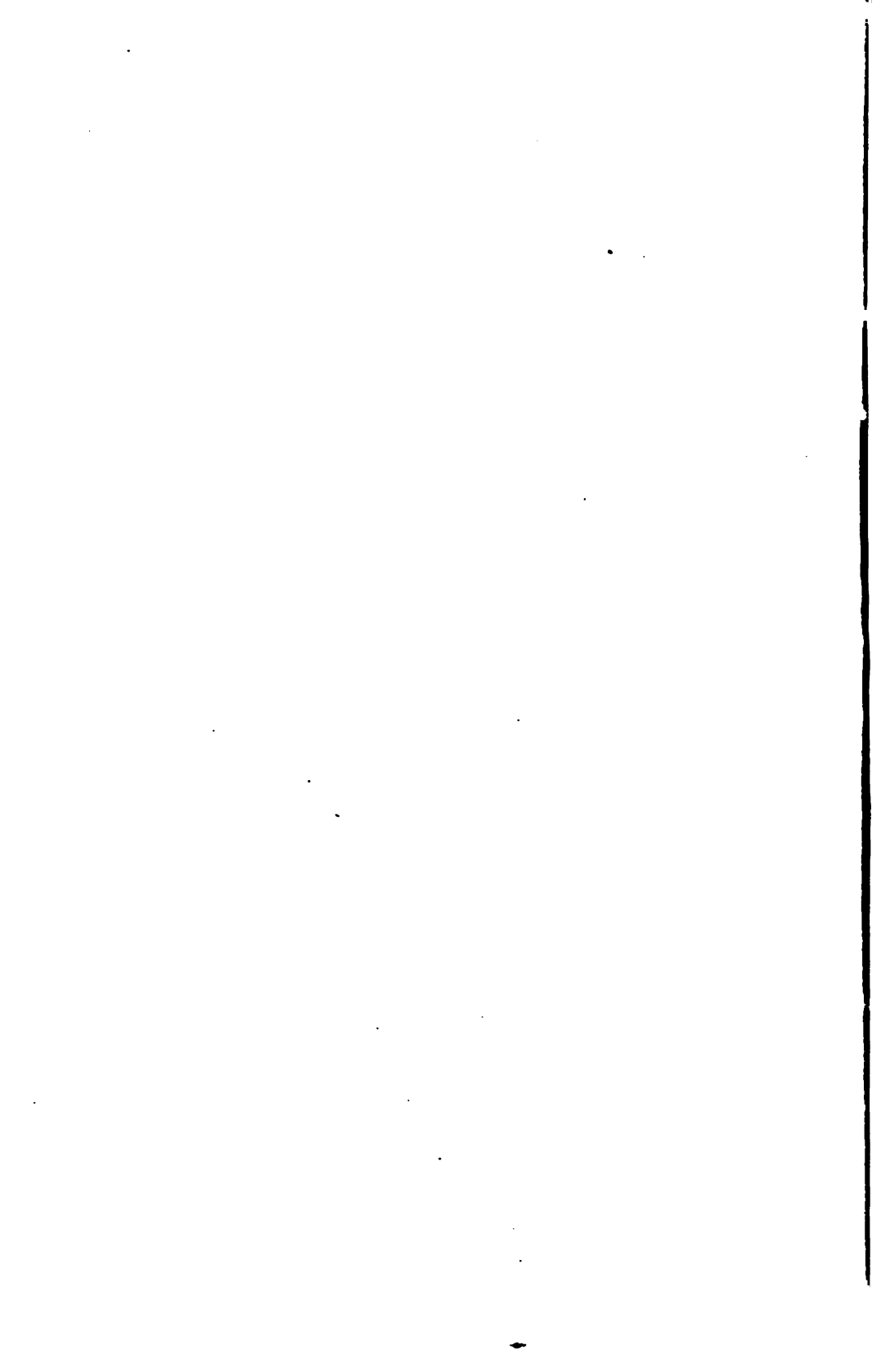
(F) Gas compressors do not induce wasteful conditions, but, on the contrary, they stimulate conservation, for the reason that, by their use, gas wells may be operated in a more economical manner.

(G) The relation of the natural or open flow to actual line flow from various wells, as determined by Pitot tube measurements, is shown in graphical form in Fig. 9.

(H) The short range of the low pressures that may be maintained in the gathering line of the gas field going to the intake side of a gas compressor is emphasized in Fig. 10.

(I) The working pressure relationship of wells discharging into intake lines to gas compressors and wells of other companies discharging into main lines not connected to gas compressors is shown in Fig. 11.

(J) 4,510 hourly pressure comparisons were made between wells of the Upham Gas Company discharging into their lines without compressors, and the wells of adjacent companies discharging into gas compressors. 90 per cent. of these showed the Upham wells to be lower than the adjacent wells, and in 26 per cent. of these the Upham wells were lower than the intakes at the gas-compressing stations drawing on the adjacent wells.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Segregation in Gold Bullion*

BY JAMES H. HANCE, E. M.,† IOWA CITY, IOWA

(New York Meeting, February, 1916)

INTRODUCTION

SEVERAL years ago the writer was connected with the Mint and Assay Service of the Federal Government as Assistant Assayer at the Salt Lake Assay Office. At that time cyanide bars formed approximately half of the bullion purchased by that office. Disagreement in valuation between the producer and the office was not infrequent and to a lesser extent between the Assay Office and the Mint, this latter issue being soon obviated, however, by adopting uniform methods of sampling the bars. A nice problem seemed to offer itself for research and the writer began a series of experiments which were soon terminated, however, by his going into other work, and which he regrets have not since been completed. The results attained may suggest lines of approach to this problem and are, therefore, offered in this article.

Frederic P. Dewey, Assayer of the Mint Service, U. S. Treasury, has had abundant opportunity to study this phase of metallurgy, and has published several articles relative thereto.¹ In the last article cited Dewey quotes analyses of cyanide bullion similar in content to that with which the writer worked, and which show some of the variations met with.

In the discussion of these papers,² comparison has been made in terms of copper bullion, but to the writer this seems of questionable value. Copper bullion may resemble silver bullion, but its similarity to gold bullion with little or no silver content is another thing. The freezing-point diagram of Roozeboom, reproduced on page 881 of the discussion cited, is for the gold-silver series as stated. The original problem under discussion, however, is *not this alloy* but gold with *base metals other than copper*. As for the statement that the corners are the worst places or

* This material in somewhat different form was submitted by the writer some years ago as thesis material in partial fulfillment of the requirements for an engineering degree from the College of Mines, University of Washington.

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¹ Frederic P. Dewey: The Assay and Valuation of Gold Bullion, *Trans.*, xl, 780 to 797 (1909); The Sampling of Gold Bullion, *Trans.*, xlv, 853 to 879 (1912); Cadmium and Nickel in Cyanide Bullion, *Engineering and Mining Journal*, vol. xciii, No. 15, 733-734 (April 13, 1912).

² *Trans.*, vol. xlv, 879-882 (1912).

positions to drill: In the writers opinion there is not much choice, if the outer film of approximately 1 in. be excluded. A glance at the diagrams accompanying this article will show that a wide variation from the dip average is apt to be encountered at any of the places drilled in these experiments.

Sampling Methods

A few details of manipulation will be reviewed here to throw light on the discussion. The large bars are cast in molds $12\frac{1}{4}$ by $5\frac{1}{4}$ in., and range from 2 to $2\frac{1}{2}$ in. in thickness. They are essentially prisms, the mold being tapered slightly to permit easy removal of the bar.

Three methods of sampling are in vogue: (1) Chipping diagonally opposite corners of the bar; (2) drilling the bar; (3) dip-sampling as the bar is being poured. Methods (2) and (3) are the ones commonly employed in sampling cyanide bullion. Drilling is done as shown in Fig. 1.

The two samples of top drillings are mixed and assayed as the top sample, and the two bottom drillings are mixed and assayed as the bottom sample. If the bars are not uniform in composition and if the variation is gradual it is readily seen that the top sample really represents an average of the top and center, or approximately a plane half way between the top and the center of the bar. Similarly the mixture of bottom drillings represents a plane half way between the center and the bottom. Now an average of these two planes is not necessarily an average of the whole bar, and where segregation is the *rule* and *not the exception*, this method may be open to criticism. This point I believe has been somewhat overlooked or neglected, and may well be worthy of experimental investigation. As emphasized by Mr. Dewey, these drillings themselves may be rather heterogeneous, the smaller fragments differing appreciably from the coarser in gold content, and this may result in wide discrepancies in assays of the same drill sample.

Dip samples may be taken as follows: One sample may be taken from the top of the melt just before it is poured into the mold, or a portion of the stream cut out just after pouring begins. This represents the top of the melt, or the bottom of the bar. A second sample taken from the pour just before completion represents the bottom of the melt, or the top of the bar. These samples are the most reliable, if the melt was thoroughly stirred just before pouring, and should check very closely within themselves and with each other.

Variations in Assays

Irregularities in the assay of cyanide bullion seemed to have been first noted by Edward Mathey,³ who thus summarizes some interesting

³Edward Mathey: On the Liquefaction of Certain Alloys of Gold, *Proceedings of the Royal Society of London*, vol. lx (1896-97), pp. 21 to 35.

and instructive experiments: (a) Alloys of gold with base metals, notably with lead and zinc, now largely met with in industry, have the gold concentrated toward the center and lower portions, which renders it impossible to ascertain their true value with even an approximation to accuracy; (b) when silver is also present, these irregularities are greatly modified. The method of obtaining "cooling curves" of the alloys shows that the freezing points are very different where silver is present and where it is absent from the alloy; (c) this fact naturally leads to the belief that if the base metal present does not exceed 30 per cent., silver will dissolve it and form a uniform alloy with gold; (d) this conclusion is sustained by experiments (described in his article) which, in fact, gradually lead up to it and enable a question of much interest to be solved.

In another series of experiments with alloys of gold and platinum, Mathey clearly demonstrates the tendency of the latter metal to liquefy toward the center of the mass while cooling.

Dewey's articles call attention to the existence of variations in assays of such bullion and the possible causes, but he does not show the nature of the variations.

Mention is made in metallurgical texts of the solvent property of silver where certain base metals, such as zinc, lead, etc., are present. The writer has had occasion to confirm this point in connection with the assay of gold bullion. If there is considerable silver present, 60 points (6 per cent.) or more, a uniform bar is generally obtained, but where the silver content is slight, segregation or liquation may take place on cooling and the resulting bar may be far from uniform in composition. With these points in mind the writer assayed a number of drillings taken from various portions of bars. Grouping these results in various ways a number of interesting and suggestive diagrams may be constructed which illustrate graphically the lack of homogeneity in some cyanide bullion.

Qualitative tests on some of the bullion indicated the presence of small amounts of mercury, lead, bismuth, arsenic, antimony, zinc and chromium. Dewey⁴ reports cadmium, nickel, iron and copper from some similar bullion. The silver content ranged from zero plus up to 10 points (or 1 per cent.) in rare instances. As a rule these bars are very brittle, and upon fracture show some crystal faces well developed, and a variety of colors. These ranged from the color of marcasite and pyrite to those of chalcopyrite. The faces of the bars were characteristically marked with small gas blebs, giving them a pitted appearance. This was especially noticeable along the upper four edges.

Now, when diagonally opposite corners of such a bar are chipped and assayed, the values obtained may have a wide variation among themselves, and in most cases are several points below the values shown by

⁴Frederic P. Dewey: Cadmium and Nickel in Cyanide Bullion, *Engineering and Mining Journal*, vol. xciii, No. 15, pp. 733 to 34 (Apr. 13, 1912).

dip samples. Hence agreement between chip samples and dips from such bullion is hopeless, and the greater the number of duplicate assays made, the more firmly established is this difference. When this feature was recognized the mint issued the following instructions for sampling such bullion: "A drilling is to be made about one inch from each edge on two opposite corners of the top of the bar, and should extend half way through the bar. These two drillings are to be thoroughly mixed and assayed as the top sample. The bottom sample is to be taken in the same manner, corners being chosen not under the top corners drilled. (This is illustrated in Fig. 1.) Dip samples are to be taken as previously described and assayed with the drill samples."

In Table I are given some results obtained from assaying such samples. Where only one assay value is recorded, as for the top dips of bar 18,

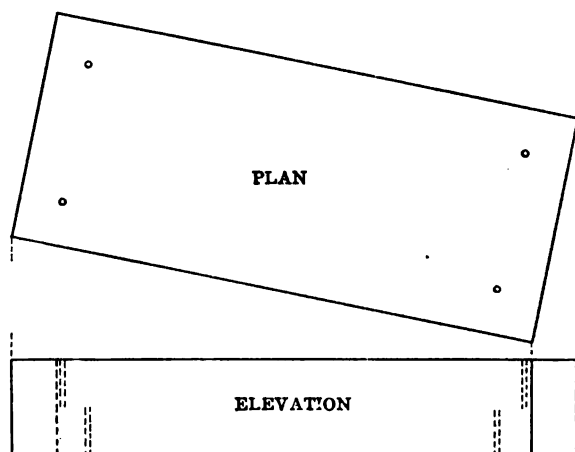


FIG. 1.

it should be understood that the other assay was spoiled, and not that it was too far off to put on record. Similarly for one of the top drills of bar 41. Eighteen of these bars were reported as having no silver (meaning less than 0.05 per cent.), the maximum silver content being 12 points (1.2 per cent.), whereas the average for the 48 bars was less than 3.8 points (0.38 per cent.).

Comparison of the assays of the top and bottom drills shows that in 38 cases the top drillings assay higher than the bottom drillings, whereas in 10 cases the reverse is true. In many instances a considerable range of assay values appears. Examining the dip assays in the same way it is found that in 33 cases the top dips assay higher than the bottom dips, and in 12 cases the reverse is true. In three bars the top and bottom dips average the same. As a rule, however, there is less difference be-

TABLE I

Bar	Top Drills	Bottom Drills	Top Dips	Bottom Dips	Average of Drills	Average of Dips
1	865.1	865.9	867.4	864.2	865.7	865.7
	865.2	866.6	867.0	864.2		
2	845.4	845.5	843.4	845.7	845.15	844.78
	844.4	845.3	845.1	844.9		
3	835.5	834.5	834.1	834.8	834.78	834.63
	835.4	833.7	834.7	835.3		
			834.5	834.8		
			834.6	834.1		
4	857.7	856.6	859.5	859.6	856.65	859.45
	856.8	855.5	859.2	859.5		
5	856.9	855.1	857.0	857.0	856.13	857.15
	856.7	855.8	857.4	857.2		
6	842.9	844.8	844.3	843.7	844.0	843.98
	843.3	845.0	844.2	843.7		
7	795.7	790.0	792.0	791.6	792.93	791.9
	796.1	789.9	792.1	791.9		
8	836.2	834.1	834.1	833.5	835.05	833.85
	835.7	834.2	834.1	833.7		
9	775.8	773.8	773.5	773.6	775.1	773.48
	776.4	774.4	773.6	773.2		
10	810.1	806.7	810.4	810.7	809.75	810.75
	809.8	812.4	811.1	810.8		
11	793.6	792.0	793.2	793.1	793.22	793.17
	792.7	794.7	793.3	792.5		
		793.1	793.2	793.4		
12	795.2	796.1	795.3	793.8	795.3	794.97
	795.3	796.5	795.2	794.7		
	796.0	792.7	794.9	795.9		
13	799.4	796.9	798.7	798.0	797.05	798.25
	799.2	797.1	798.6	798.0		
	796.9	802.8	797.7	798.5		
14	928.4	928.0	929.5	928.9	928.25	929.13
	928.5	928.1	929.1	929.0		
15	834.2	831.4	832.8	832.3	832.98	832.63
	834.1	832.2	833.0	832.4		
16	854.5	853.2	853.5	853.3	853.98	853.43
	854.7	853.5	853.4	853.5		
17	849.4	847.5	847.6	847.3	848.48	847.5
	849.2	847.8	847.6	847.5		
18	898.4	898.6	898.9	898.8	898.15	897.94
	898.3	897.6		898.4		
		898.5		897.6		
		897.5		898.0		
19	788.3	787.9	788.8	788.9	788.03	788.93
	788.1	787.8	789.0	789.0		
20	803.7	802.8	803.4	802.7	803.2	802.87
	803.6	803.7	803.1	802.6		
	802.5	802.9	802.0	803.2		
21	796.4	794.3	794.6	795.1	794.82	794.87
	794.5	794.0	795.2	794.8		
	793.7	796.0	794.9	794.4		
22	875.1	874.7	876.0	874.9	874.9	875.53
	875.4	874.4	875.9	875.3		
23	813.3	813.1	814.5	814.0	813.26	814.09
	813.2	813.0	814.3	813.5		
	813.2	813.3	813.8	814.5		
	813.1	813.9	813.4	814.7		

TABLE I.—(Continued)

Bar	Top Drills	Bottom Drills	Top Dips	Bottom Dips	Average of Drills	Average of Dips
24	813.1 812.3 812.5 812.9	812.7 812.8 812.2 812.3	813.5 813.1 813.3 812.7	812.8 812.8 813.4 813.4	812.6	813.13
25	806.4 806.8	806.8 806.3 806.2 806.6	807.3 807.2	806.9 806.8 806.8 806.6	806.52	806.93
26	875.3 875.0	873.9 873.4	874.8 875.0	874.3 874.2	874.4	874.68
27	772.7 772.8	773.7 773.5	774.1 774.3	773.7 773.6	773.18	773.93
28	811.6 810.4	807.8 807.7	806.8 808.9	808.1 807.9	808.88	807.93
29	772.2 772.1	769.6 770.3	771.9 772.1	771.1 770.9	771.05	771.5
30	758.1 757.9	755.5 754.7	757.8 757.8	756.7 756.6	756.55	757.25
31	743.6 743.0	743.6 743.3	742.7 742.6	741.6 741.7	743.38	742.15
32	875.6 875.5	874.3 874.2	875.7 875.4	875.2 875.0	874.9	875.33
33	886.8 887.2	886.7 886.6	887.8 886.7	886.5 886.6	886.83	886.9
34	860.3 859.6	858.7 858.5	858.6 858.5	858.2 858.4	859.25	858.43
35	868.2 868.8	868.2 868.4	868.1 868.0	868.0 868.0	868.4	868.03
36	885.7 885.1	884.2 884.1	884.1 884.0	884.0 884.0	884.78	884.03
37	887.0 886.8	885.5 885.6	885.3 885.4	885.5 885.7	886.23	885.48
38	890.2 889.5	888.8 888.7	889.2 889.1	889.2 889.1	889.3	889.15
39	888.5 888.7	889.5 889.5	889.7 889.9	889.9 889.7	889.05	889.8
40	896.1 896.1	895.6 895.5	895.9 896.0	895.7 895.7	895.83	895.83
41	903.8 903.6 903.6	901.0 901.1 901.0 901.2	901.8 901.6 902.0 901.8	901.4 901.5 901.9 901.8	902.2	901.36
42	892.3 890.4	888.5 888.6	888.7 888.9	888.8 888.7	889.95	888.78
43	893.8 894.8	892.5 892.9	892.0 891.0	892.8 892.7	893.5	892.13
44	939.8 940.2	938.9 939.1	938.7 938.9	938.7 938.8	939.5	938.78
45	935.1 935.2	933.7 933.1	933.5 933.5	933.3 933.3	934.28	933.4
46	942.0 941.7	941.2 941.5	941.1 941.2	941.3 941.3	941.6	941.23
47	694.3 694.3	704.0 703.9	698.4 699.5	699.6 699.4	699.13	699.23
48	854.2 853.8	853.0 852.8	851.2 847.8	853.0 852.8	853.45	851.2

tween the top and bottom dips than there is between the top and bottom drillings.

Comparing the assays of drillings and dip samples it is seen that in 28 instances the drillings assay higher than the dips; in two cases they are alike, and in 18 cases the dips assay higher than the drillings. In nearly every instance the drillings show greater variation than do the dip samples, and yet in some samples (Nos. 1, 2, 12, 20, 28, 30, 31, 41, 43, 47, and 48) there is a marked variation here, even in dips from the same portion of the melt.

The molds are of cast iron and are heated to about 150° to 200°C. before the bullion is poured into them. As the metal is a better heat conductor than the air, that portion of the bullion which is in contact with the mold probably solidifies first, whereas the top cools last. This might produce a bar of the following characteristics: Since the four lower corners chill first, they might contain a higher percentage of the metals and non-metals of the highest freezing points. By the term "freezing point" as here used is meant a resultant of the absolute freezing point of the metal or alloy, and the saturation point of this same metal or alloy in the melt. This is offset to a certain degree by some of the metal solidifying as soon as it strikes the mold and not permitting any segregation. Next in order would be the four upper corners, then the bottom and sides, and lastly, the top which may be somewhat protected or blanketed by a thin coating of slag. The middle of each face would probably freeze later than any other point of that face, and the center of the bar, approximately, would solidify last, or possibly a thin film through the center and parallel to the bottom might be the last to take the solid form. The specific heats of the various elements making up the bar would exert a differential tendency in its cooling. If mercury were present, on account of its low melting point, it might tend to segregate to that portion of the bar which freezes last. The solvent properties of the metals and their alloying characteristics would also exert an appreciable influence here. Arsenic is known to cause a lack of uniformity, even where present in small amounts, but the exact nature of this action does not seem to be understood. A considerable amount of the volatile and easily oxidized elements are driven off during the melting stage, zinc, lead and bismuth being detected in the flues, but as charcoal covers are used, this expulsion is not complete. Unfortunately, an elaborate qualitative and quantitative analysis of this bullion was not feasible, owing to lack of equipment, and to certain restrictions placed upon the operating departments of the assay offices. Extensive drilling was therefore resorted to for a few bars, and some interesting variations determined.

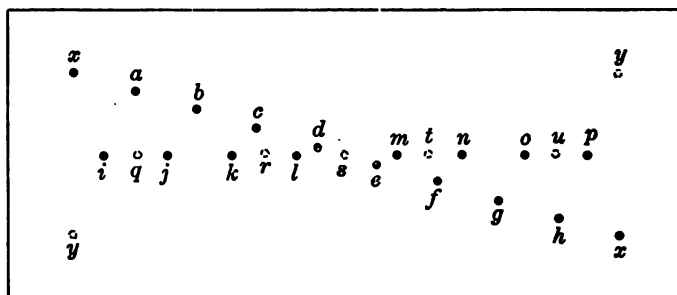
From the drillings and dips taken in the usual manner for bar 49, the following assay values were obtained:

TABLE II

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
881.7	881.1	881.6	881.2	Less than one point of silver present.
881.2	881.1	881.4	881.3	

This bar was specially drilled as indicated in Fig. 2, each drill hole extending to the middle of the bar. The bar was $10\frac{1}{2}$ in. long, $4\frac{1}{2}$ in. wide and $2\frac{1}{2}$ in. thick, and weighed 824.81 troy ounces.

The points x, x represent the regular places for drilling the top of the bar, and y, y the places for taking bottom drillings. The points $a, b, c, d,$



Plan

1"
Scale

FIG. 2.

$e, f, g, h, i, j, k, l, m, n, o, p$ are on top of the bar and spaced as shown. The points $q, r, s, t, u,$ are on the bottom of the bar. The drillings from

TABLE III.—Assay Values of Special Samples of Bar 49

A	B	C	D	E	F	G	H
881.9	881.6	881.6	882.8	882.6	882.3	884.1	883.0
881.7	881.3	881.5	882.6	882.9	882.9	883.3	883.1
I	J	K	L	M	N	O	P
883.8	879.0	882.4	882.2	883.2	882.7	884.1	883.8
885.9	883.2	881.6	882.3	881.4	881.8	883.6	884.2
Q	R	S	T	U			
885.7	884.7	880.9	880.5	882.5			
884.4	882.2	880.1	881.4	882.3			

each hole were carefully mixed to insure uniformity (as far as possible) and were assayed in duplicate. Bars 50 and 51 were drilled in the same manner, and all the samples likewise assayed in duplicate. The values determined from these assays are given in the accompanying tables.

TABLE IV.—*Assay Values of Special Samples of Bar 50*

A	B	C	D	E	F	G	H
876.8	876.0	877.2	877.1	875.4	875.8	874.6	875.3
877.6	876.6	877.2	876.3	876.1	875.5	875.4	875.3

I	J	K	L	M	N	O	P
878.1	877.2	878.2	877.4	874.8	876.5	877.6	877.8
877.4	877.3	878.1	876.4	875.8	875.9	877.6	878.5

Q	R	S	T	U
877.4	875.6	874.0	875.1	876.6
878.5	875.5	873.2	875.5	877.1

From the drillings and dips taken in the usual manner for Bar 50, the assay values shown in Table V were obtained.

TABLE V.—*Assay Values for Bar 50, Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
874.3	874.4	874.1	874.2	Two points of silver present.
874.0	874.4	874.4	874.4	

Bar weighed 822.09 oz. Dimensions same as Bar 49.

TABLE VI.—*Assay Values of Special Samples of Bar 51*

A	B	C	D	E	F	G	H
883.2	884.2	883.5	882.7	883.3	885.1	885.4	884.9
883.4	883.3	884.6	885.3	883.6	884.6	886.1	884.1

I	J	K	L	M	N	O	P
887.1	886.1	885.1	884.5	883.5	884.9	887.4	885.6
887.1	886.3	884.7	884.7	883.4	884.7	886.9	885.7

TABLE VI.—(Continued)

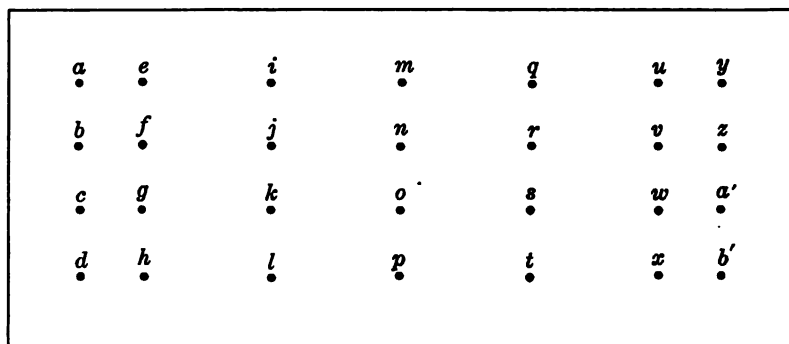
Q	R	S	T	U
888.7	883.9	883.1	883.7	884.6
888.7	882.8	882.3	883.1	885.7

From the drillings and dips taken in the usual manner for Bar 51, the assay values shown in Table VII were obtained.

TABLE VII.—Assay Values for Bar 51 Samples Taken in Ordinary Manner

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
883.0	882.2	882.2	882.8	Five points of silver present.
883.8	882.8	882.3	882.9	

Bar weighed 815.31 oz. Dimensions same as Bar 49.



Top Plan, Bar 52

Scale $\frac{1}{2}$ "

FIG. 3.

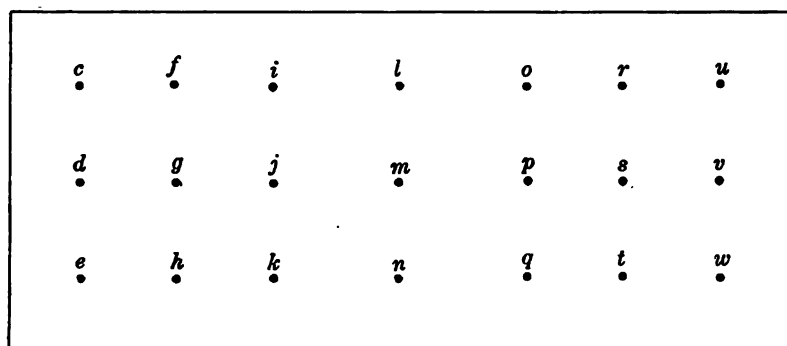
A large number of drillings were taken from two other bars. The method of drilling is shown in Fig. 3. The dimensions of the bars were: length, $12\frac{1}{4}$ in., width, $5\frac{1}{4}$ in., and thickness, $2\frac{1}{4}$ in. Bar 52 weighed 949.34 oz., and Bar 53 weighed 942.34 oz.

Drillings from *a* to *b'* through *z*, inclusive, were taken from the top of the bar. The bottom drillings *c'* and *d'* were taken under *y* and *d*, respectively. After assaying samples *a* and *b'* separately, the samples were carefully mixed and assayed as the top drills. Similarly, *c'* and *d'* were first assayed separately and then mixed and assayed as the bottom drills. Separate treatment of the end drillings shows different values for the two ends of the bar, a point lost sight of where the drillings are first mixed and then assayed. This becomes more noticeable when diagrams are constructed on these values.

TABLE VIII.—*Assay Value of Special Samples of Bar 52*

A	B	C	D	E	F	G	H	I	J
838.2	840.8	839.9	839.4	838.1	840.0	840.8	838.9	838.4	838.7
838.0	840.5	839.6	839.1	838.0	840.3	840.4	836.7	837.6	838.7
K	L	M	N	O	P	Q	R	S	T
837.5	837.9	839.5	838.8	838.5	839.3	837.6	837.6	838.9	837.6
838.0	838.8	839.4	838.6	838.5	838.7	837.3	837.9	837.4	836.8
U	V	W	X	Y	Z	A'	B'	C'	D'
837.7	840.0	838.4	839.2	835.5	838.9	838.3	838.2	836.5	837.4
838.6	840.4	839.9	839.3	836.4	838.5	839.3	838.4	836.2	837.4

From the top drillings obtained by mixing samples *a* and *b'*, the bottom drillings obtained by mixing samples *c'* and *d'*, and the dips taken in the usual manner for Bar 52, the assay values given in Table IX were obtained.



Bottom Plan, Bar 53

Scale 1"

FIG. 4.

TABLE IX.—*Assay Values for Bar 52 Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
837.9	836.9	837.5	838.0	Six points of silver present.
838.2	836.8	838.2	837.6	
838.3	838.4	837.9	838.9	
837.8	838.4	838.4	838.8	

Drillings *c* to *w* inclusive (Fig. 4), were taken from the bottom of the bar. Samples *a* and *b* were drilled from the top of the bar over *e* and *u* respectively. After assaying samples *a* and *b* separately, the samples were carefully mixed and assayed as the top drillings. Similarly, *c* and *w* were first assayed separately and then mixed and assayed as the bottom drillings. As was true with Bar 52, comparison of assays *a*, *b*, *c*, *e*, *u* and *w* shows considerable variation between the two ends of the bar.

TABLE X.—*Assay Values of Special Sample of Bar 53*

A	B	C	D	E	F	G	H
830.8	831.8	831.0	831.6	831.8	830.2	831.0	830.9
831.7	831.2	831.4	832.5	831.7	830.3	830.9	831.3

I	J	K	L	M	N	O	P
832.3	831.3	830.4	830.3	830.2	831.1	830.9	830.2
831.0	832.1	830.9	830.1	830.6	830.2	831.0	830.1

Q	R	S	T	U	V	W
829.3	829.3	833.2	829.0	829.9	831.5	828.8
829.7	829.4	832.8	828.8	829.0	831.1	828.7

From the top drillings obtained by making samples *a* and *b*, the bottom drillings obtained by mixing samples *c* and *w*, and the dips taken in the usual manner for Bar 53, the assay values given in Table XI were obtained.

TABLE XI.—*Assay Values for Bar 53, Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
831.3	831.2	831.0	830.0	Six points of silver present.
830.5	831.4	831.1	829.7	
832.3	829.8	829.3	831.5	
.....	829.6	831.7	831.5	

Construction of Diagrams

The horizontal length of the diagram in each case represents the outside dimension of the bar along the line of the profile. As no sample is taken less than an inch from the edge of the bar, the curves all terminate inside of the diagrams. The vertical control is that of fineness, the

extremes taken so as to include all assay values on the bar. This feature is apparent on the last or composite diagram for each bar, upon which are assembled the assay values of all the *drillings* from that bar. In this diagram the values are all projected upon a plane passing lengthwise through the center of the bar and normal to the end and bottom faces.

The light solid horizontal line passing through the diagram—and curves in *some cases*—represents the average value of the dip assays, and probably is a very close approach to the true value of the bar. The

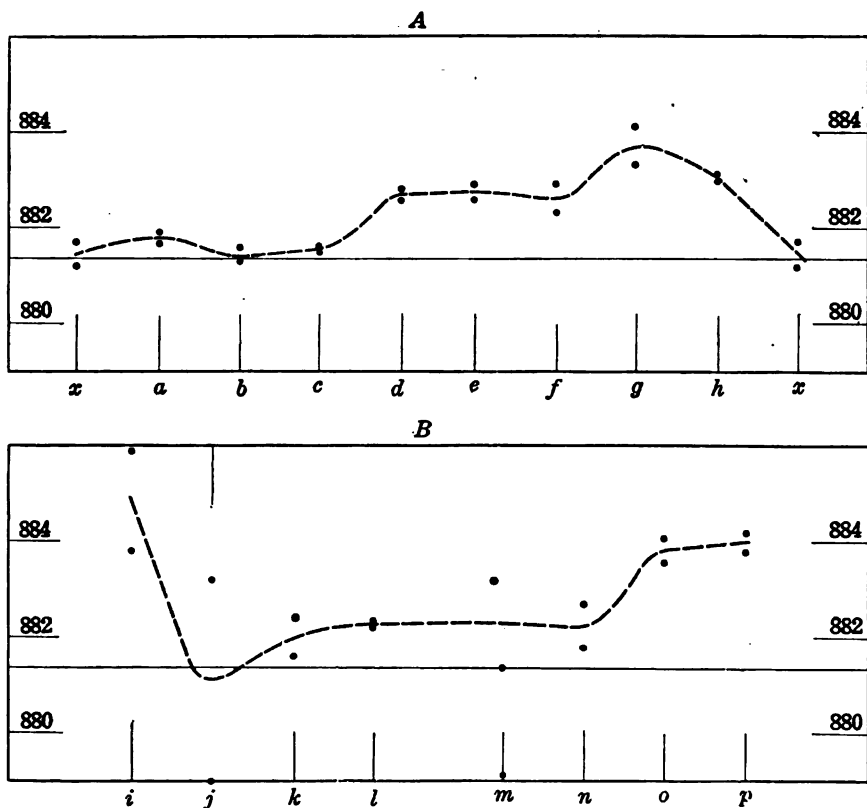


PLATE I.—BAR 49.

letters along the base of the diagram are the points on the bar from which the drillings were taken, and over these letters are placed the corresponding assay values. Each diagram is thus an assay profile along a line through the lettered points, the last diagram for each bar being a composite profile along a longitudinal medial plane, all assay values of the *drillings* being projected upon this plane. All *drilling* assay values are placed upon the diagrams except Plates XII and XIII, and the curves are drawn through the average for any point.

Several methods of drawing curves through these average values are possible, and each one may be nearly equally justified.

1. Straight lines may be drawn between consecutive average values, the profile consisting of a zigzag line made up of straight segments.

2. A smooth curve may be drawn through consecutive average values, without regard to what might lie beyond each end of the curve.

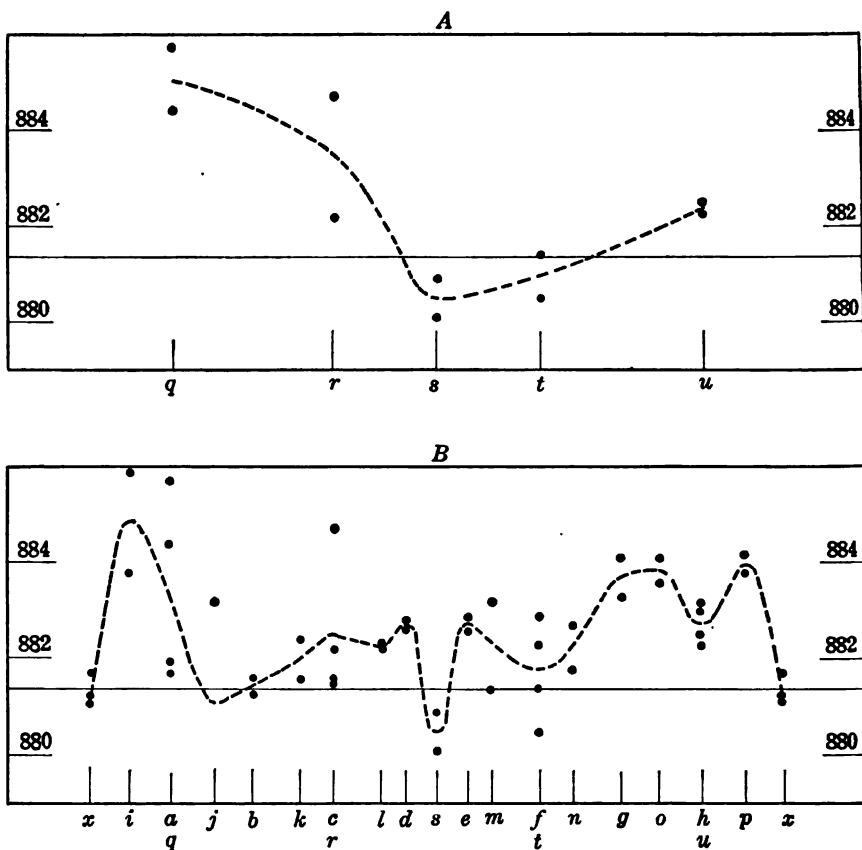


PLATE II.—BAR 49.

3. A smooth curve may be drawn through consecutive average values, and this curve be bent toward the horizontal at each end.

4. A smooth curve may be drawn through consecutive average values, but warped toward the average dip value. This would assume all other than dip values as departures from the true value and as abnormalities.

5. An average of two consecutive averages may be taken and its departure from the dip average noted (call this departure d). Now take the dip average as a datum line, and half way between the two averages considered (horizontally), insert d with its sign changed. This is merely

a device for obtaining low values to offset the high values. Either short straight lines or a smooth curve may then be drawn through all points, average values and the new points thus computed.

The first method assumes that assay values were taken at all maximum and minimum points, and that the gradation between points is uniform. This is certainly not necessarily true, and is probably never attained.

The second method considers that all maximum and minimum points are represented, but does not suggest uniform gradation between points sampled. Thus the curve between points is dependent upon adjacent

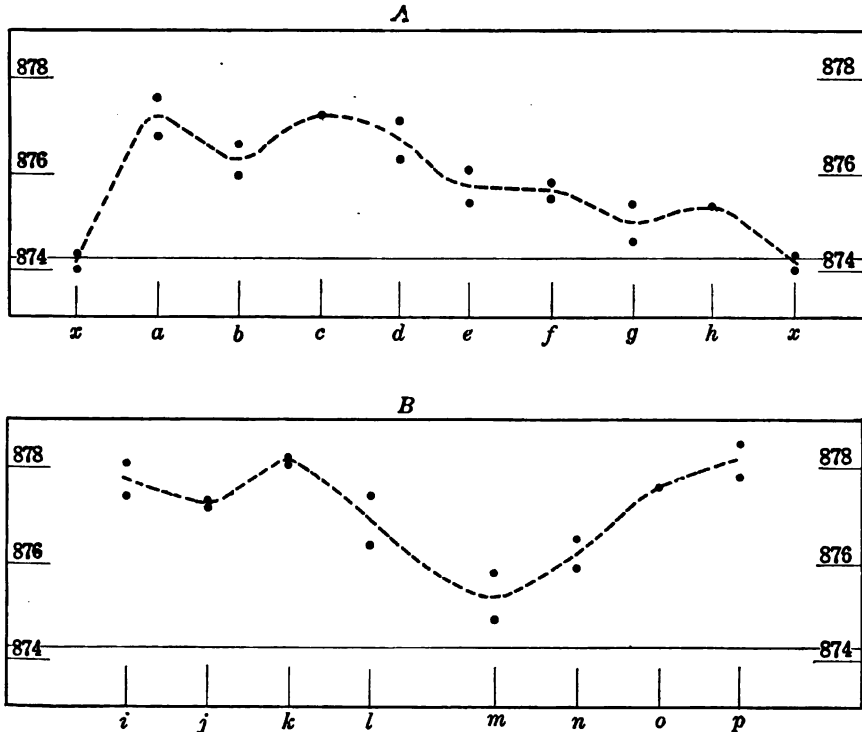


PLATE III.—BAR 50.

values as well, and therefore is probably in less error than the zigzag line of method (1). As it is not necessarily true that all maxima and minima are represented, there is undoubtedly a departure from actual conditions, but the only remedy for this would be a determination of essentially all points of the bar, a proceeding manifestly impossible.

The third method bends the curve toward the horizontal at each end. This undoubtedly possesses merit in some cases, but may introduce errors in other instances. It is essentially the same as (2).

The fourth method brings out interesting contrasts, but certainly

cannot represent true conditions in the bar. It might still leave all or most of the profile above or below the dip average, a lack of balance probably not the case. A glance at the assay values and diagrams for bars 49, 50, and 51 will show the difficulty of realizing such conditions with some bars.

Such a diagram as outlined in the fifth case or method would show some of the variations demanded if we are to consider the dip average as the true value. Although introducing mid-points, in the opinion of the writer it may more nearly meet the conditions of the case than any of the other

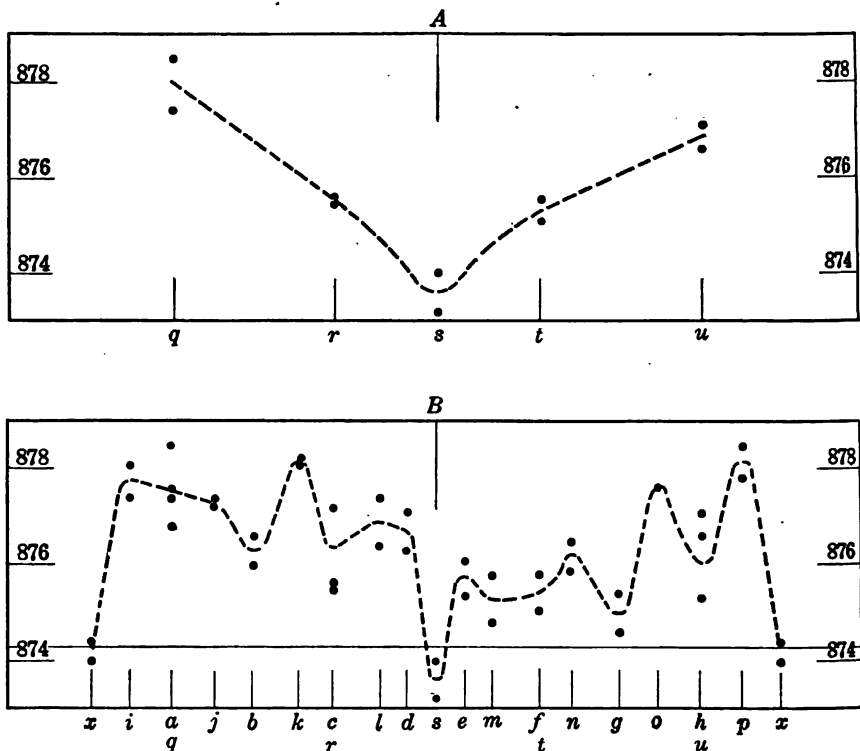


PLATE IV.—BAR 50.

methods. In all probably it does not properly place these variations, but at the same time it shows what the minimum variations must be if the dip value is the true datum line. For instance, such a composite diagram as Plate VI, B cannot tell all the story, for where are the low values to offset the high ones and give the true bar value? Much of this adjustment is undoubtedly accomplished in the outer film of the bar, outside the curve terminals, but not necessarily all.

In the first nine plates the second method was used. For bar 53 the third method was followed, and it shows curves nearly identical with

those obtained with the second method. In Plate XIII the first and fifth methods were used.

Discussion of Curves

Bars 49, 50 and 51.—As these three bars were of the same size and shape, and as they were sampled in identically the same manner a comparison of their diagrams may be instructive.

Noting the curves along the diagonal line on top of the bar it is seen

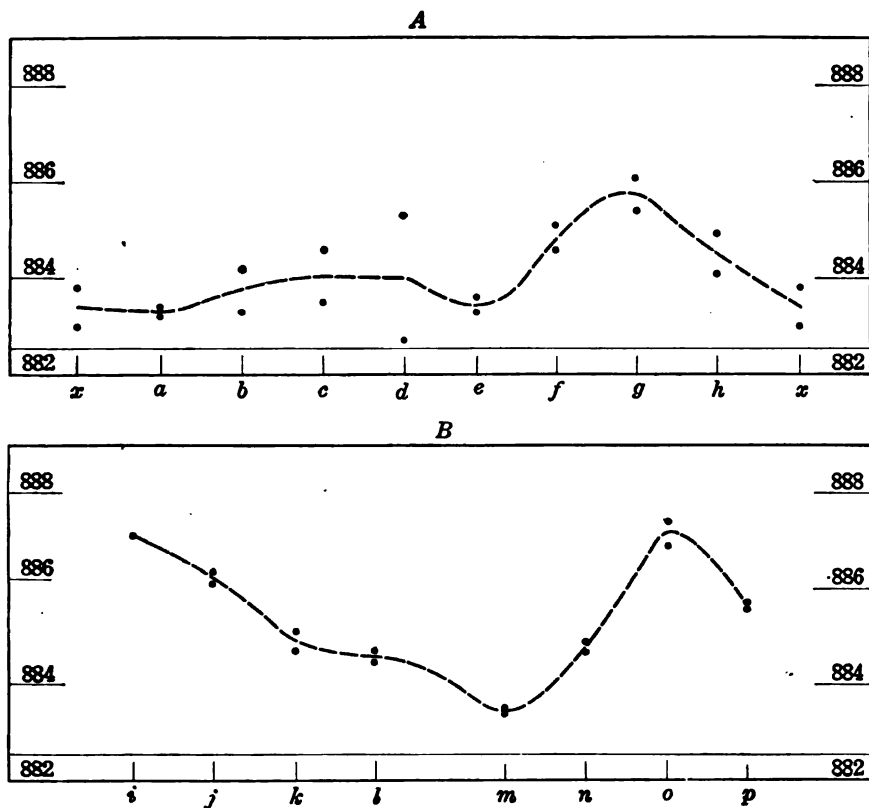


PLATE V.—BAR 51.

that essentially all the *average* values are above the average dip value, emphasizing the difficulty of obtaining the fineness of such bullion from drillings. Another feature is the large bulge or wave-like crest in the curve near one end. The reason for this is not known, as the melt is poured across the mold from end to end, and principally at the center. As the drillings from each end are mixed and assayed in duplicate, both end values are alike. Duplicate values would not likely result if the drillings from each end were assayed separately. In the case of each of these three bars the curve is convex upward.

Examination of the curves for assay values along the central line on top of the bar shows a similar wide range of values and a still greater departure from the average dip value. Each of these curves is essentially concave upward, in contrast to the convexity of the curves along the diagonal line. In speaking of the cooling of the melt, mention was made of the probability that the center of the top of the bar may be the last part of this face to solidify. If this is true in principle for the bar itself,

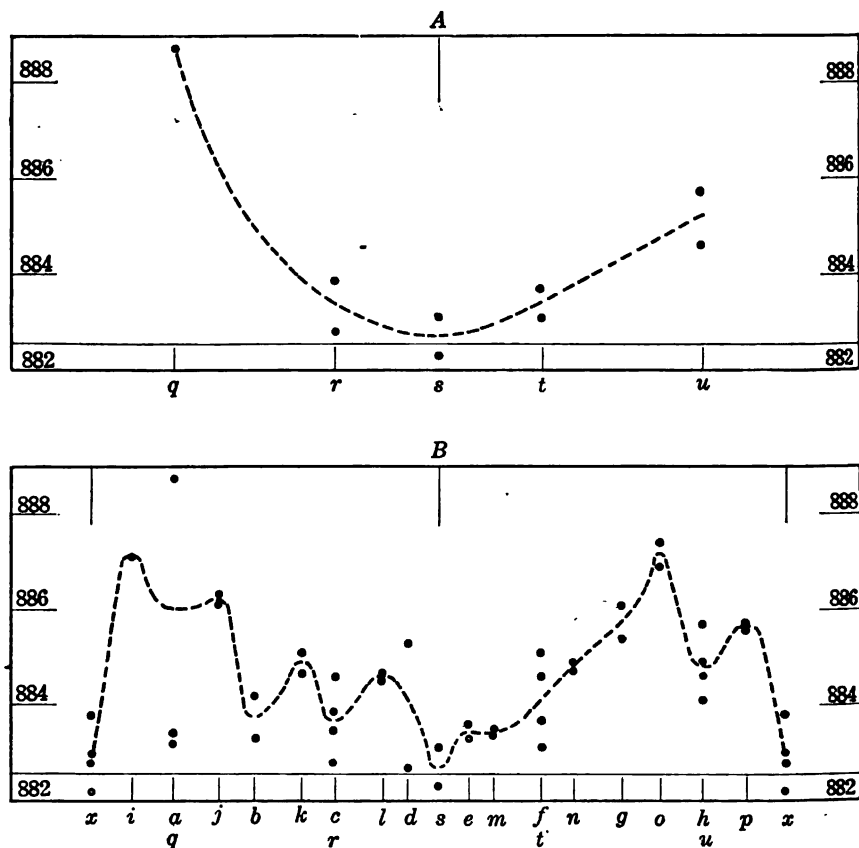


PLATE VI.—BAR 51.

and a metal such as mercury were present, a lower assay value for this portion of the bar might be expected. However, it might be expected to fall below the average dip value rather than above. As in the previous case, nearly all the assay values lie *above* the average dip value.

The curves for the central line along the bottom of the bar have one noticeable trait in common; namely, a concavity upward which is due to a lower value near the center of the bar. In keeping with this feature is the hypothesized action due to the unequal rates in cooling. The

center of any face probably cools or solidifies later than any other portion near the edges, such as along the borders or near the corners. At the centers, these curves, except that on Plate VII, *A*, pass below the average dip value, although most of the curve is above.

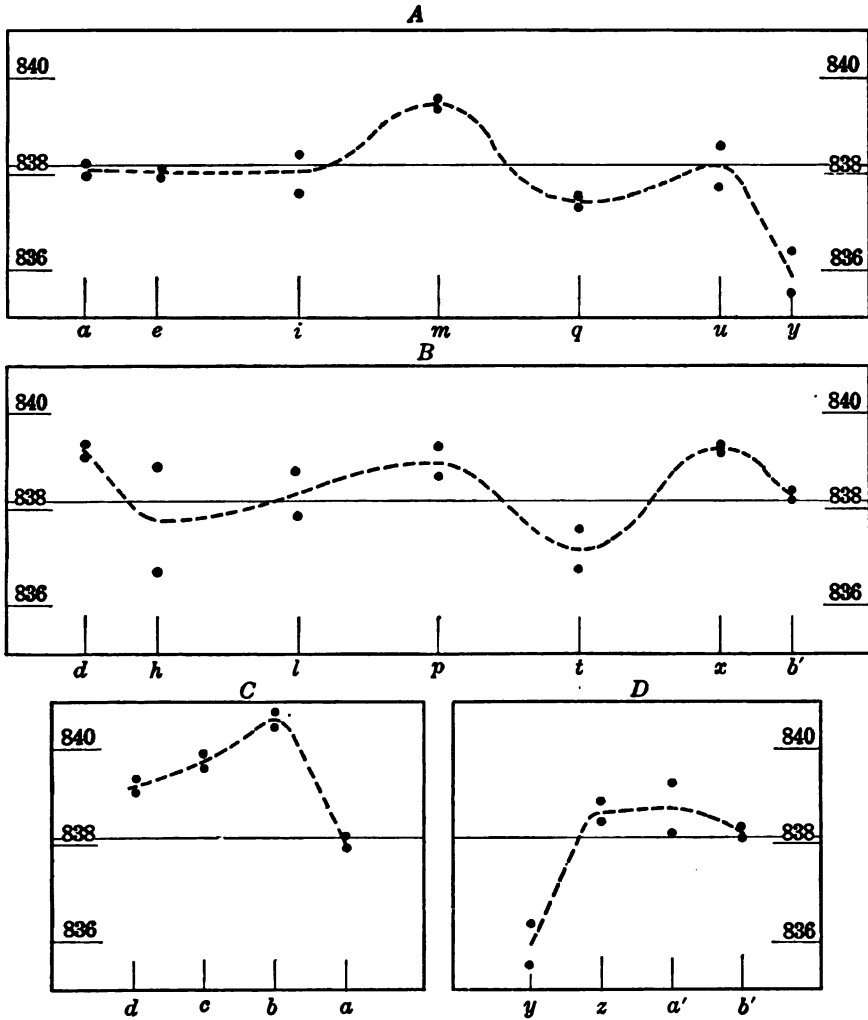


PLATE VII.—Bar 52.

In the composite diagrams the curves are still more irregular but all illustrate the high values of drillings and the inadvisability of attempting to determine fineness of such bullion from such samples.

In Plates II, *B* and IV, *B* the end averages are nearly coincident with the dip averages. In Plates VI, *B*, IX, *D* and XII, *B*, however, these end

average values are not the same as the dip averages, but are higher in some cases and lower in others.

Bars 52 and 53 were sampled according to a different scheme. Lines were drawn parallel to the edges, and drillings were taken at short intervals along these lines (see Figs. 3 and 4).

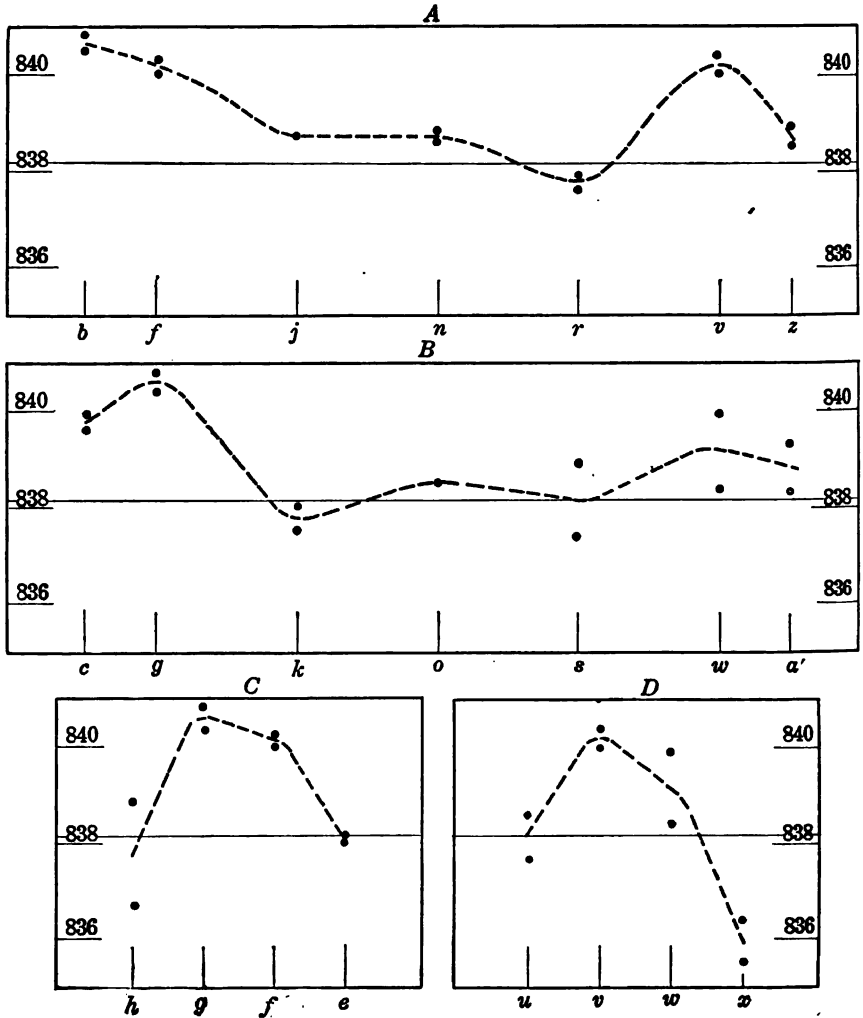


PLATE VIII.—BAR 53.

The bottom of bar 52 and the top of bar 53 were sampled in the official manner previously described. Plotting these values lengthwise and crosswise of the bar, two sets of curves were obtained. A composite diagram is also given for each bar. The diagrams are arranged in pairs so as to contrast similar portions. Thus for bar 52 the profile along

aeimquy corresponds in position to that along *dhlptzb'*. Each curve connects a series of averages for assays of drillings taken parallel to, and at a given distance from the edge of the bar. On a given face these profiles

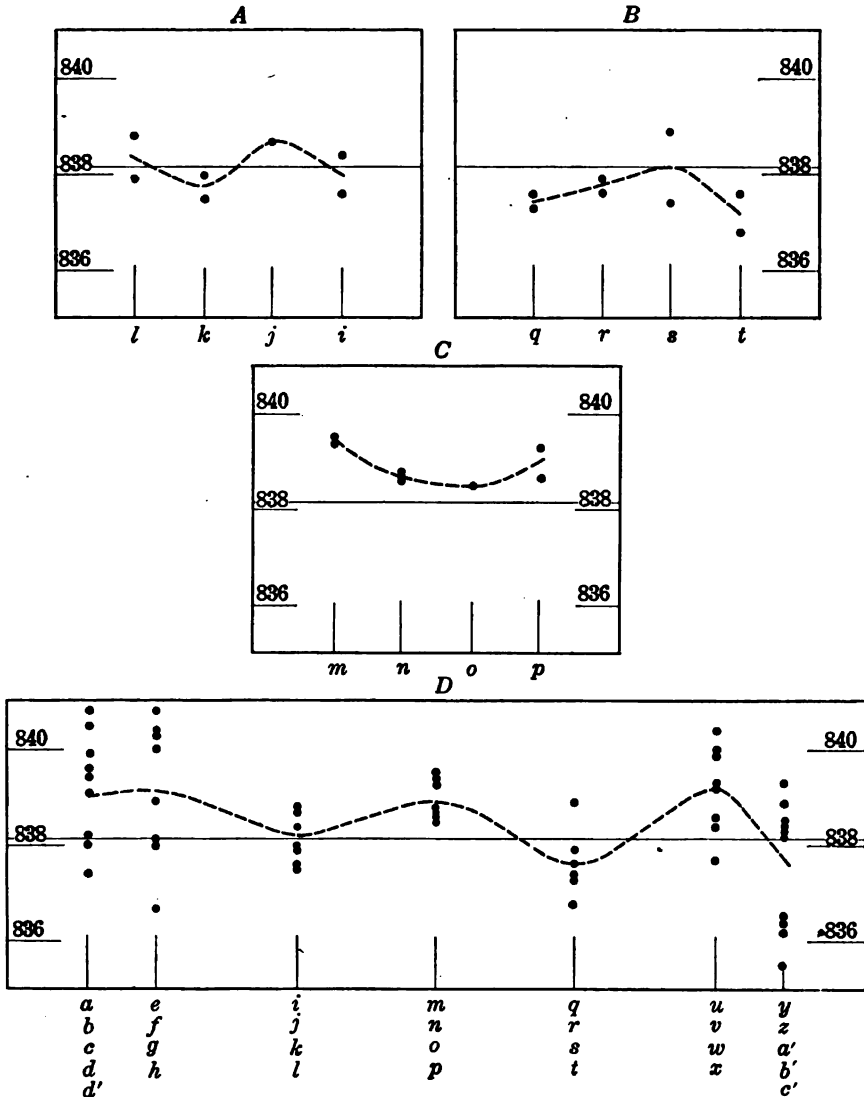


PLATE IX.—BAR 52.

might be expected to show similar variations if the liquation in such cooling is merely a matter of rate of cooling rather than a complex function depending upon many variables, conspicuous among which is apparently composition. Possibly the term liquation is here inapplicable, as the

solid bar may (and in all likelihood does) somewhat resemble a rock of heterogeneous mineral composition, rather than a liquated magma. Close proximity to the walls of the mold may exercise much the same function as proximity to inclosing wall rock in the case of a cooling magma. Here, however, the solution of metal in metal (or non-metal)

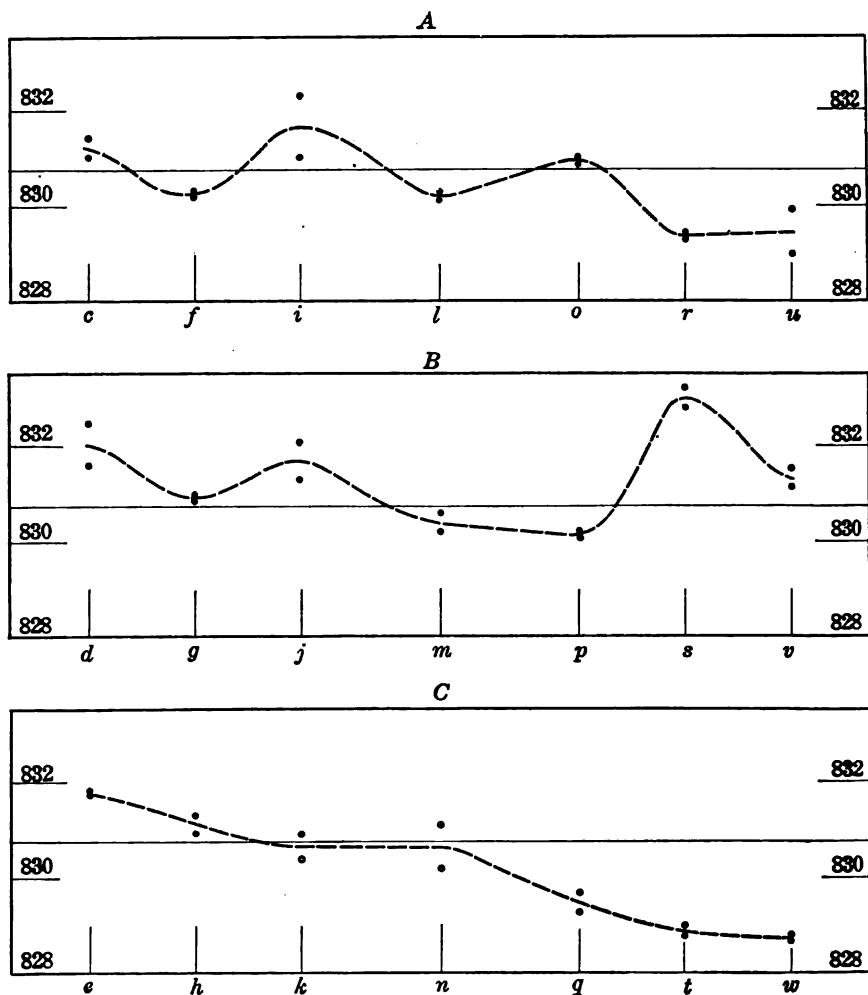


PLATE X.—BAR 53.

and the peculiar properties of alloys may vary the final products considerably.

Thus in Plate VII, *A* and *B*, each curve shows a conspicuous crest at the center and a conspicuous trough to the right of this crest. Or the right half of the curve shows two crests with an intervening trough. In

each case the central crest separates two dissimilar curves. Neither curve is decidedly concave or convex upward. Unlike preceding curves they are approximately bisected by the dip average.

Comparing *C* and *D* a slight similarity is noted, in that both are convex upward and the variation from one edge of the bar to the opposite

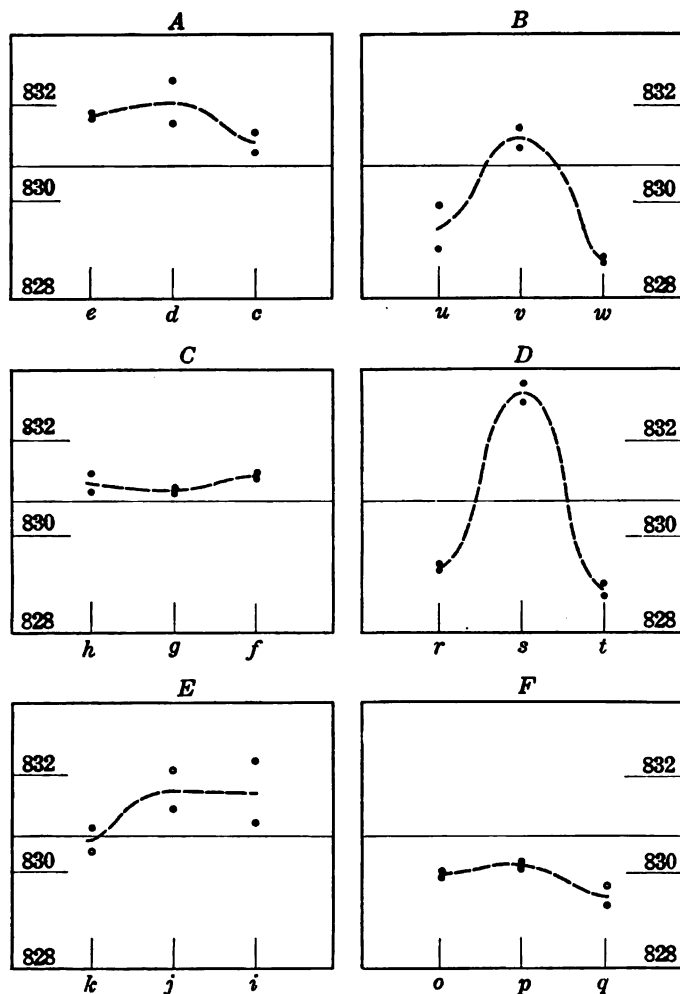


PLATE XI.—BAR 53.

side is consistent for both profiles. Considerable variation, however, is shown in these profiles.

In Plate VIII, contrasting *A* and *B*, some like features are apparent. The curves are both concave upward and average well above the dips. The central trough in each reaches nearly the same minimum value, whereas the ends show similar maxima.

C and *D* are both convex upward and average well above the dips. These curves which are convex upward, are just the opposite of *A* and *B* which are concave upward, and show that if there is such a thing as concentration of a metal such as mercury toward the center, it is more than offset by some reverse process, at least locally.

In Plate IX, *A* and *B* are not much alike. *A* is unlike *C* on Plate VIII, although it is an adjoining profile, whereas *B* is similar to *D* of Plate VIII, but less pronounced. *C*, Plate IX, is concave upward and

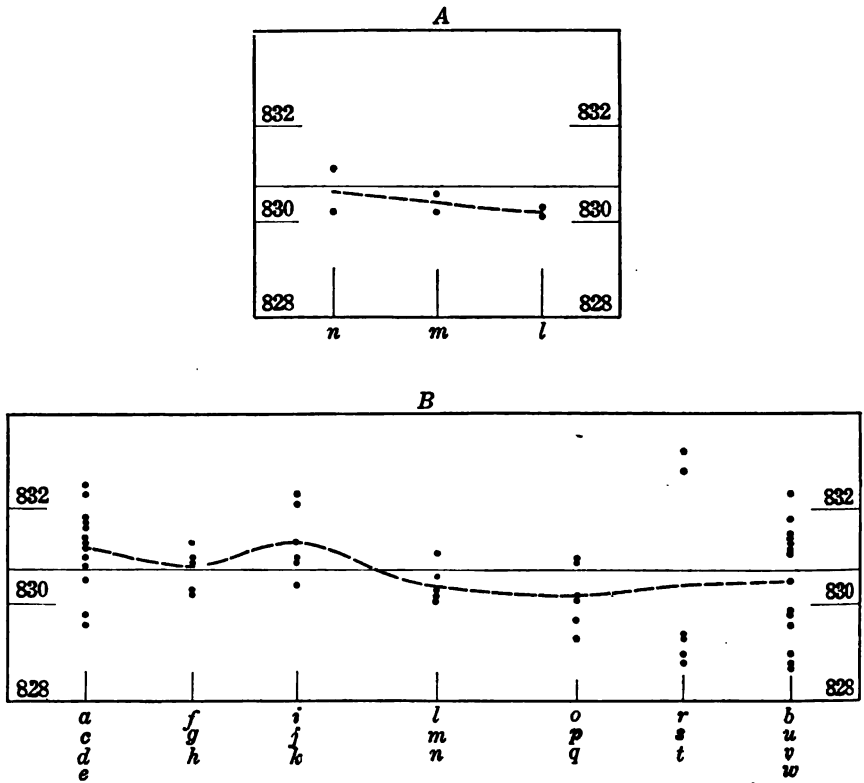


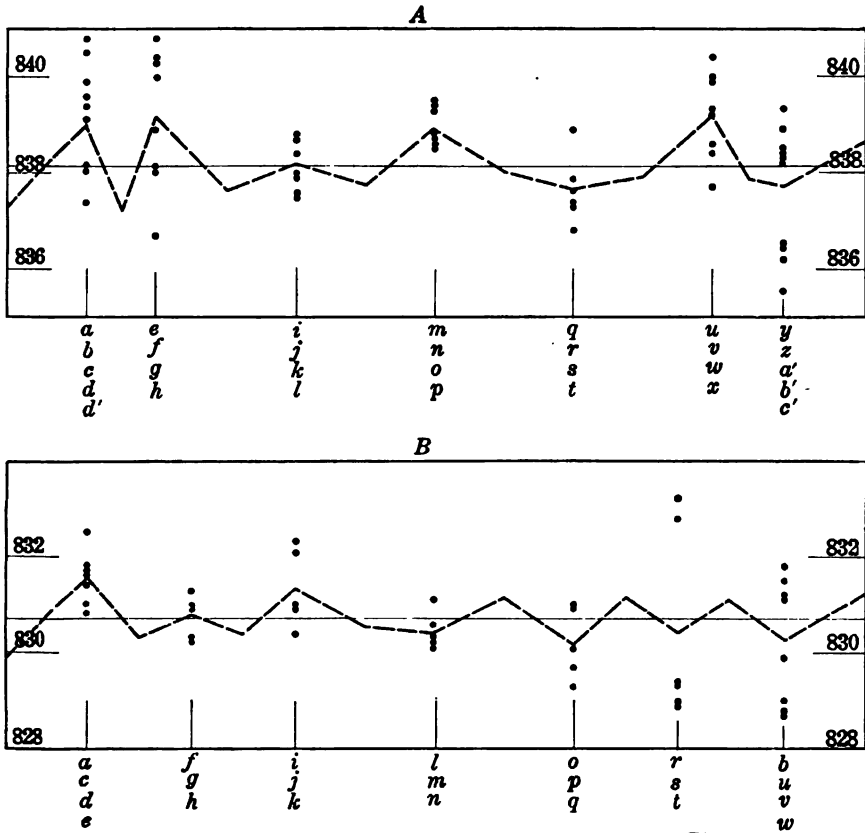
PLATE XII.—BAR 53.

in harmony with a possible concentration of such a metal as mercury at the center. At variance with this, however, is the fact that the entire curve is above the dip average.

The composite diagram *D* shows a curve with three well-defined crests and two troughs, and with an average somewhat above the dip average. Although the range in individual values is considerable, the point averages are fairly close to the dip average. The grouping of the assay values shows well-defined maxima and minima points.

Bar 53 was drilled in a way somewhat similar to bar 52 except that the drillings were from the bottom of the bar instead of the top. This, it was hoped, might furnish data for contrasting cooling in a face exposed to the air or to a thin slag coating, as compared with the cooling of a face next to the mold. Drilling one or several of the lateral faces might show other gradations and bring out interesting contrasts.

In Plate X, A and C are similarly situated, and both show downward



Mixed Top and Bottom Drills not Included on these Diagrams

PLATE XIII.—BARS 52 and 53.

gradients from left to right. Neither one, however, is decidedly concave or convex upward, and the curves are not very similar. Curve A shows erratic tendencies whereas C indicates a less variable decline in value from left to right. B, which is along the medial line of the bottom of the bar, is concave upward, but is in decided contrast with both A and B. In some ways, however, it seems to be a composite of both A and C, except for the values of s and v which seem abnormally high for this bar.

In Plate XI most of the curves are convex upward, but do not go

together well in pairs. For instance, *A* and *B* are similarly situated, but are in strong contrast; similarly for *C* and *D*, *C* being slightly concave upward. Since *E* is on the same side of the bar as *A* and *C* we might compare them. It is not very far from being a composite of *A* and *C*, although the left end is rather low.

Comparing *F* with *B* and *D* we find that the convexity upward of *B*, which is more conspicuous in *D*, has nearly disappeared in *F*, the maximum point having fallen well below the dip average.

A of Plate XII is a fair composite of *E* and *F* on Plate XI, and lies between them on the bar. It is nearly a straight line, the variation being a slight concavity upward. *B* on Plate XII is the composite for bar 53, and although it shows wide range between individual assays, indicates a bar more nearly uniform than any of the preceding. The curve is slightly concave upward although the average value of drillings is very close to that of the dips.

In the diagrams on Plates X, XI and XII, the curves are bent toward the horizontal at the ends, but this feature is inconspicuous in the final product.

If a curve were constructed according to the fourth method suggested at the beginning (warping all portions of the curve between the maxima and minima toward the dip average) the result would be rather striking and extremely improbable. This would be especially conspicuous for bars 49, 50 and 51.

For these reasons two diagrams are given for composite values of bars 52 and 53, constructed according to the first and fifth methods, the zigzag line of short straight segments being used rather than the smooth curve. The composite diagrams for both of these bars show average points which are less extreme, measured from the dip average, than those for bars 49, 50 and 51, and for these reasons were chosen. The intermediate and extreme points are chosen so that the areas above the dip average but below the profile are equal to the areas below the dip average but above the profile. Or considering the areas between the dip average and the profile as positive or negative according as they are above or below the former, the positive areas equal the negative. Some such balance must exist if the dip average represents the true assay value of the bar. Such a diagram does not show, necessarily, the exact location or manner of such variations, but it does give a minimum variation. Possibly much of this balance may be accomplished in the outer inch of the bar, rather than within this profile. Complete adjustment in this outer portion, however, seems improbable to the writer, especially in profiles such as Plates II, *B*, IV, *B* and VI, *B*. In profiles such as Plates IX, *D* and XII, *B* where values in the profile are both below and above the dip average, it seems much more reasonable to suppose that an approximate balance is maintained within the profile. The same may be true for Plates II, *B*,

IV, *B* and VI, *B*, in which cases the local variations are extreme in places. Although the gradation is almost surely not along straight lines, the turning points may be as sharp as those in the diagram, and are not necessarily curved crests or troughs.

In both profiles on Plate XIII one end of the diagram shows a rising value, whereas the other end shows a decreasing value. Neither extreme is far from the dip average but the erratic variation is thus shown. These two bars, however, are less heterogeneous, or variable through a smaller range, than many cyanide bars, and although an average of all the drillings is close to that of the dips in these two cases, it is decidedly otherwise in many other instances, such as bars 49, 50 and 51, as well as many of the bars listed between 1 and 48.

Another interesting feature brought out by these diagrams is the variation in assay value of individual samples. In the simple diagrams this variation seems equally conspicuous along the profile (or bar) and does not appear to favor any single portion of the bar. The composite diagrams cannot of course be here considered, as there is superposition of points, due to projection on the medial plane.

In a study of these curves a few characteristics seem to be common, but no attempt will be made to offer other than tentative reasons for such. It is to be hoped, however, that a further detailed examination may be undertaken to clear up these points. Such an examination is of necessity confined to the government mints (or possibly assay offices) but for some reason the men in these places have been handicapped by a lack, either of time or of a desire for such an investigation. Considerable extra time is required for such detailed sampling, and still more elaborate methods must be pursued before a satisfactory solution is reached. This is out of the question in the assay offices as the time element is an important feature, and also on account of the bookkeeping system used, no sampling being possible after settlement on a bar is made. At the mints an unusual opportunity is afforded, and such work would be of considerable value to the service as well as of scientific interest.

Summary

1. Drillings taken near an edge are extremely variable, but generally assay higher than the dips, a fact which may be due to early solidification and an accompanying segregation in those portions of the bar of metals of the highest freezing points, taking into account mutual solubilities and alloying tendencies. In a platinum-gold alloy this same characteristic is very apparent, but is due, as Mathey observes, to the liquation of the platinum.

2. The center of the bar, top and bottom, is usually lower in value than the dips. As suggested early in this paper, this may be due to a late solidification in these and similar points, which results in a higher per-

centage of base metals with low freezing points. Mutual solubilities may be a factor here, also, but the substance in large excess (gold) does not concentrate toward the center or last place of solidification as does quartz in the cooling of a quartz-rich magma.

3. In a majority of cases the top drillings assay higher than do those of the bottom, and the same is true of the dips. In order to balance the personal equation the method of weighing up assays was reversed so that the weigher who received the top samples at first, later received the bottom samples. For this reason samples were frequently exchanged in weighing. The same variations, however, continued to be apparent.

An extreme range in the dip assays is in some cases due to the presence of impurities introduced while the dips are being taken. Where a scorifier is used to catch part of the stream of metal, small pieces of clay might easily be caught and retained in the bullion granules.

4. Drillings taken at any one point along these profiles vary as widely in assay value as drillings of any other point similarly chosen. Also, maxima and minima points are as apt to occur near the center of the bar as near the ends. From this standpoint the corner drillings seem no better nor worse than drillings from nearer the center.

5. The liquation in some cyanide bars is similar to that which takes place when the rare metals are present in appreciable amounts, but in the former instance it seems to be due to the presence of certain base metals.

Attention is again called to the fact that these difficulties are not met with where a small amount of silver is present, as the presence of this latter metal seems to diminish segregation. Lessened segregation is probably due to the solvent properties of silver toward certain base metals such as lead, zinc, etc. Silver may exert an appreciable influence on other metals or non-metals also. Where 90 or 100 points of silver were known to be present in the bar, no dip samples were taken, and the drillings rarely failed to check well within the limits allowed. This is doubtless due to the fact that silver or silver-gold alloy in which the silver content is nearly 10 per cent., or better, is a good solvent for the metals or non-metals causing the segregation. Pure or silver-free gold is not a good solvent for these substances and hence does not prevent the formation of an irregular bar.

This suggests one way to overcome the difficulty of assaying cyanide bullion, although it does not solve the problem. If to all such cyanide bars known to contain little or no silver, a small amount of the metal be added in melting, a uniform bar after melt could be easily obtained and its fineness accurately determined. From the fineness thus obtained the inquartation silver could be subtracted, leaving the silver fineness of the original bar. As such gold bars are alloyed with silver for refining operations, the above suggested operation would mean a little additional work for the computers only, but this extra work would be more than offset by the increased accuracy of the gold assay.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Brown-Coal Mining in Germany

BY GEORGE J. YOUNG,* B. S., MINNEAPOLIS, MINN.

(New York Meeting, February, 1916)

DURING the spring of 1910 I visited a number of open-pit brown-coal mines and underground workings in the vicinity of Halle, Halberstadt, Leipsic, Cologne and Bonn. The notes which I took and the observations which I made at the time have been condensed and make up the principal part of this paper. Certain supplementary information has been taken from several sources which are given in the paper.

The mining of brown coal is one of the important industries of Germany. The coal varies in color from dark brown to almost black and is soft like loam. Pieces of wood and strips of bark and even parts of tree trunks are frequently encountered in the excavation although the coal for the most part consists of a pulverulent, more or less matted mass of small particles of vegetal matter. It is a product between peat and lignitic coal. An analysis, taken from *Coal Resources of the World* shows the approximate composition of the material: Moisture, 53.73 per cent.; ash, 4.98; fixed carbon, 18.08; volatile carbon, 23.31 per cent. The principal characteristic is the amount of moisture. This seldom runs less than 48 per cent. The ash is usually low and ranges from 5 to 10 per cent. The sulphur content is usually less than 0.5 per cent. The heating effect of the fuel ranges from 2,000 to 2,500 calories.

The commercial deposits are confined to the Tertiary period although

	Reserves in Millions of Tons	
	Actual	Probable and Possible
Prussia and North German States.....	6,069	3,676
Saxony.....	3,000	
Bavaria.....	75	293
Hesse.....	170	99
Total.....	9,314	4,068

* Professor of Mining University of Minnesota.

some deposits, not of commercial importance, occur in the Pleistocene epoch. In Saxony and Thuringia, which produce about 47 per cent. of the German output, the deposits are found in the Eocene-Oligocene division of the Tertiary. Most of the other deposits are found in the Miocene division. The distribution in Germany is given in the preceding table:¹

The important centers about which brown coal is mined are Bonn, Cologne, Halberstadt, Halle, Leipsic, Semphenburg and Brunswick.

The deposits present a great variety in shape and size. They range from relatively thin beds, 9 to 30 ft. thick, up to great basin deposits



FIG. 1.—BROWN-COAL PIT NEAR BORNA.

100 to 300 ft. in thickness. They are contained in clays, sands, sandstones, marls; in some instances limestone beds are intercalated. They are more or less erratic in distribution and extent. Many of them are covered by glacial drift consisting of fine sand and light gravel. This may be only a thin layer from 20 to 25 ft. thick, or may range from 75 to 300 ft. in thickness. Faulting was not conspicuous in the pits visited with the exception of one in the vicinity of Borna, south of Leipsic. The latter deposit was trough-shaped, the main axis of the trough being cut by four transverse faults. Fig. 1 shows the edges of two of the blocks. The non-conformable glacial drift above the inclosing formations and the flat topography of the country south of Leipsic are also shown.

¹ From *Coal Resources of the World*, vol. i, p. 89.

Open-Pit Methods

Wherever the thickness of the overburden permits, the brown coal is won by stripping the overburden and excavating the coal by modifications of the "milling method." In most of the pits visited the overburden consisted of glacial drift, remarkably free from boulders, composed principally of sand and to a less extent of clay. The flat nature of the topography requires the excavation of the pit below the general level of the surface and greatly facilitates the placing of the surface plant and the arrangement of the tracks for the handling of the spoil.



FIG. 2.—CONTINUOUS-BUCKET EXCAVATOR FOR REMOVING OVERBURDEN.

The overburden is excavated by a continuous-bucket excavator of a type practically unknown in the United States (Fig. 2). The Parsons trench excavator is quite similar in principle. The buckets of the excavator are attached to a pair of heavy chains which are supported by two sprocket wheels and rollers, the latter in turn being supported by a steel frame or ladder. The ladder is pivoted at the upper end so it can be raised and lowered through an angular range from horizontal to 45° below horizontal. The arrangement is somewhat similar to the ladder and bucket run of a continuous-bucket dredge. The construction

is much lighter and the chain of buckets is operated in the reverse direction as compared with the dredge, that is, the empty buckets, mouth down, descend on the upper side of the ladder while the loaded buckets are drawn up on the lower side. As the buckets pass around the upper sprocket wheel the contents are dropped into either a hopper or a metal chute. The discharge hopper or chute is set at a sufficient height to allow the spoil cars to pass beneath. The digging ladder and driving mechanism are integral with a platform supported by trucks which run on a double pair of rails. The distance between the two sets of tracks is sufficient to admit of the placing of another track upon which the spoil cars may run beneath the loading platform. The excavator can be driven by its own power over a train of spoil cars, filling the cars as it goes, or the train of cars can be shunted into position by a light locomotive.

While there are many modifications of the excavator above described I only saw two types in operation. One is called the *tiefbagger* (deep excavator) used in making a cut below the level of the track upon which the machine operates; the other is the *hochbagger* (high excavator), used in making a cut above the track level of the machine. The machines are made for different capacities and depths of digging. The following table will give the range of sizes and capacities of the machines of this type manufactured by the Lübecker Maschinenbau-Gesellschaft:

TYPE

	B	A	C	F	L
Capacity in 10 hr. digging time, approximate cubic yards:					
Easy soil.....	2,400	1,800	900	400	220
Medium soil.....	2,000	1,500	700	300	170
Hard soil.....	1,600	1,200	500	200	120
Deepest cut, feet.....	45	30	24	15	15
Horsepower (approximate).....	90	50	38	16	12
Coal consumption in working time, pounds....	3,860	3,200	1,100	880	660
Capacity of buckets, cubic feet.....	8.5	6.4	3.5	1.8	1.25
Cost of machine { From	\$11,250	\$8,750	\$6,250	\$4,500	\$2,750
F.o.b. Lübeck { To.....	13,000	10,250	7,500	5,500	3,500
Shipping weight, metric tons.....	70	48	34	22	12
Number of men required.....	2 to 3	2 to 3	2 to 3	1 to 2	1

Approximate cubic yards represent cubic meters in the original table; table taken from H. Bansen's *Die Bergwerksmaschinen*, vol. ii, p. 35.

The excavator employed in the pits visited makes a cut about 25 ft. deep and leaves a face sloped to 45° or less as conditions require. In operating, tracks are laid along the line of the cut; when the cut has been completed the tracks are shifted back and a new cut taken. Spoil

is handled in small cars hauled by light locomotives. The spoil dumps are built up on the surface or backfilled in the pit. They present no particularly novel features.

From one to three cuts or benches are sometimes necessary in stripping a deposit. As soon as the coal is exposed a steep incline is excavated in the coal and extended until a working face from 50 to 100 ft. in vertical height is obtained. From the floor of the pit thus established, drifts of small cross-section are driven at intervals of 50 ft. into the base of the bench. At intervals of 25 ft. along the course of a drift, chutes are constructed to the surface. At the mouth of each chute a crater is started



FIG. 3.—OPEN-PIT MINING SHOWING STEEP "CRATERS."

and the coal worked into the chute. The coal is excavated by hand, the worker using a single-pointed pick and cutting the coal from the bottom of the crater upward. Footholds are cut in the sides of the crater, and where extremely steep ropes are used to prevent falls. The steepness of the craters is well illustrated in Fig. 3. The crater is worked out until the sides reach a slope on which the coal fails to slide. Drifts extend from the open floor of the pit radially or transversely along an axis of the pit. The ribs which are left between the drifts are excavated from the floor of the pit. By using a movable chute mouth, constructed of wooden planks and braced by inclined diagonal struts against a vertical face of coal, a large proportion of the coal in the rib can be won without shoveling.

The stump of the rib and the coal excavated in squaring up the face preparatory to moving the chutes must be shoveled. In excavating thin portions of a bed the movable chute is used instead of the drifts and the mill holes.

Drifts are constructed as small as possible and the back is arched so that they will be as near self-supporting as possible. Little timber is used and that principally at the point where the vertical chutes connect. The cars used are from 28 to 30 in. wide and $3\frac{1}{2}$ ft. high above the rail. The car is round bottomed, 21 cu. ft. in capacity, constructed of steel; it is dumped by means of dumping cradles. The chute gates are simple

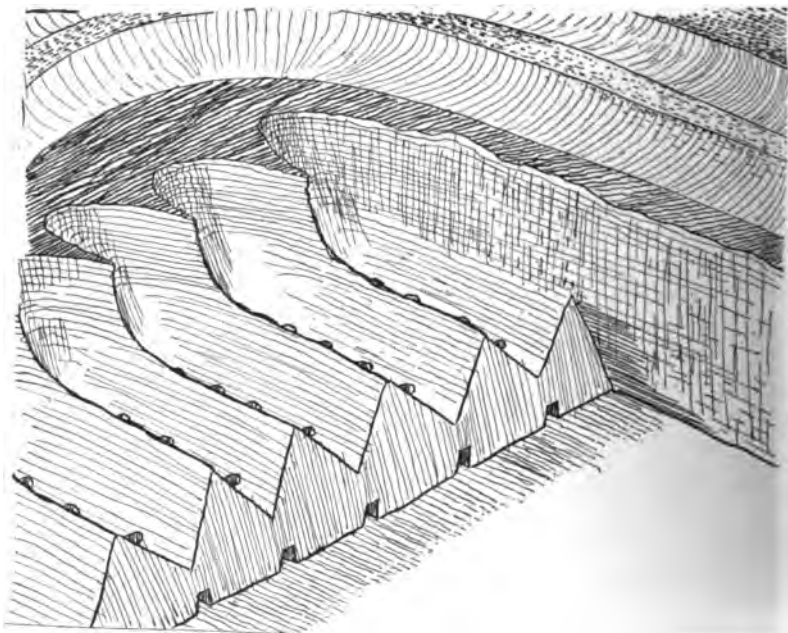


FIG. 4.—APPEARANCE OF A PIT IN WHICH A MECHANICAL CUTTER WAS USED TO MINE AND LOAD THE COAL.

in design, consisting of two or three boards held by cleats on the side boards of the chute.

The general layout for opening up the pit depends upon the shape and size of the deposit. I noted two general systems used on the larger and thicker deposits. In one the initial cut is started at the foot of the main incline and is extended parallel with and on the longer axis of the deposit. This can be done by starting three or four parallel drifts from the foot of the incline and extending them parallel with this axis. Mill holes are developed and follow up the drifting. The ribs are taken out and the floor of the pit cleared. At right angles to the first cut, parallel crosscuts are extended at intervals of 50 ft. and as rapidly as the flanking walls of

the initial cut permit. Craters are then started in the flanking walls. In the other system the initial cut is made transversely to the main axis. From the pit floor thus developed, drifts are driven parallel to the main axis of the deposit. The line of advance of the craters is parallel to the main axis. In one working near Bonn this method of opening the pit was observed. The coal in the ribs was won by means of a mechanical cutter which not only cut the coal from a face which was continuous across the pit but also loaded it into cars. The appearance of this pit and the face developed by the mechanical cutter is shown in Fig. 4. The



FIG. 5.—PIT IN OPERATION SHOWING SYSTEM OF TRACKS AND THE "MILLING" METHOD IN USE.

sequence of the stripping influences to a certain extent the method used in laying out the pit and usually favors the method first described.

The transportation problem is of interest as it involves methods which have probably been developed in this form of mining. The problem consists in the handling of a large number of small units or cars which are loaded at a number of points and which must be moved to a common dumping point. The cars are hand-trammed from the drifts and crosscuts over relatively short lengths of track which may converge at a common point or which may be parallel and terminate in a common track transverse to the drifts. A double track equipped with chain haulage serves the secondary tracks or the common points. The chain, consisting of

4-in. links, is driven by a sprocket wheel and gears at the end of the run. Motor drives are customary. A plate in which is cut a V-shaped slot projects upward from the end of each car and engages with the links of the chain. The chain is supported by the cars which are spaced at 20 to 40-ft. intervals, but is carried around corners on guide sheaves. At each turn the track is graded so that the cars gain speed on the chain and disengage (the chain is elevated sufficiently at the turns to accomplish this), gravitating around the turn where they are picked up by the sag of the chain and hauled along the next tangent. One or more runs may be necessary in a given pit. Loads are taken out on one track and the empties returned on another. The incline giving access to the pit is served by a chain also. The system is almost automatic and delivers the loads to the tipple and returns the empties with but little manual assistance

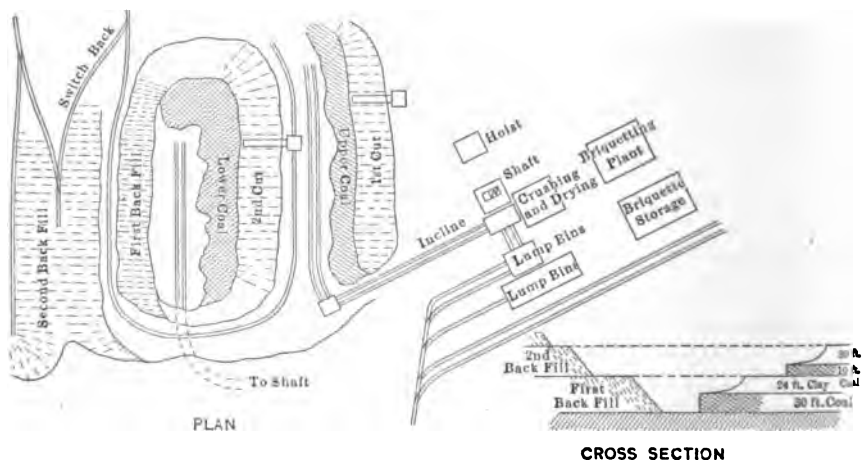


FIG. 6.—OPEN-PIT IN WHICH TWO COAL BEDS ARE BEING WORKED.

except at the points where the cars are attached and the empties removed. The *kettenban* (chain railroad) works on the same principle as the continuous-rope haulage system. The system as a whole made a favorable impression upon me.

In one of the smaller pits near Halle the upper portion of the overburden, consisting of sand and soil, was stripped by hydraulicking and backfilled into the lower part of the pit. Retaining dams were put in on the toe of the spoil slopes, the sluices discharging back of them. The remainder of the overburden, consisting of moderately compact shales and clays, was removed in two benches by undercutting, caving, and shoveling by hand into cars. The cars were hauled in trains by horses to the backfill.

Drainage in the pits is a comparatively simple problem. Electrically driven pumps connect with a sump in the lowest part of the pit and are operated when necessary.

Fig. 5 shows a pit in operation and illustrates the system of tracks and the milling method in use. Fig. 6 is a sketch plan of an open pit in which two beds were being worked.

Underground Methods

The conditions which limit the underground methods are: Relatively low selling price of the coal, extensive thick deposits, soft spongy material,

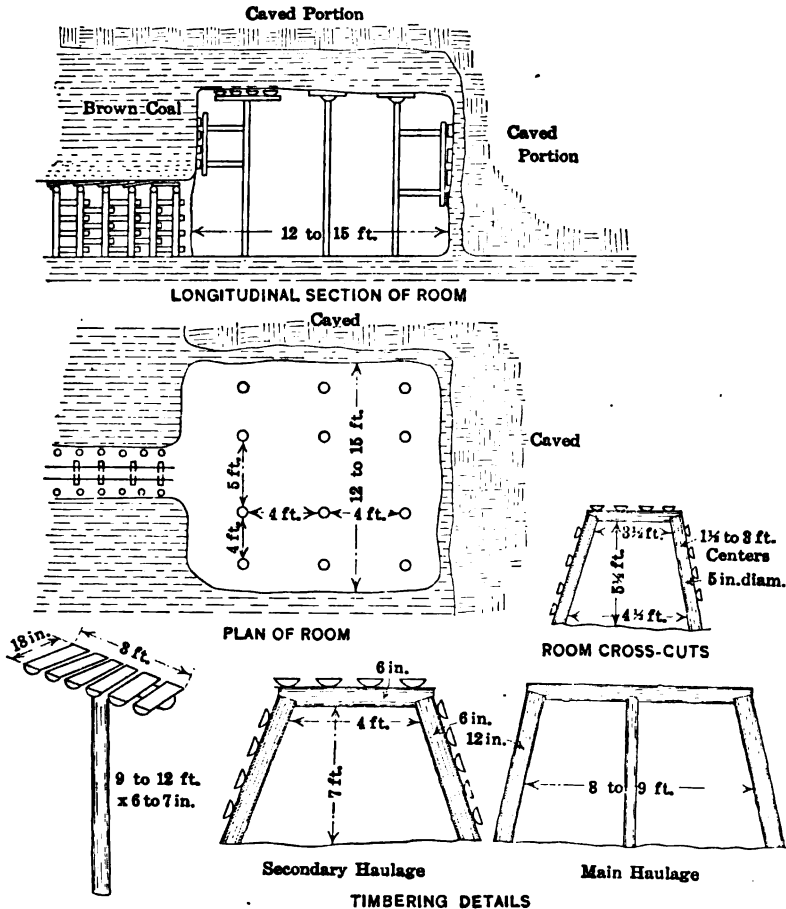


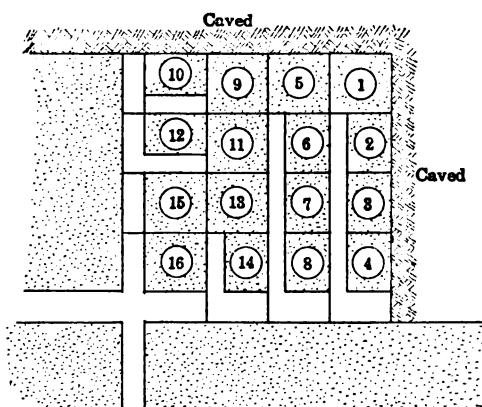
FIG. 7.—UNDERGROUND MINING SYSTEM.

a cover of sand and soft sedimentaries, depths ranging from 200 to 300 ft. and water ranging from moderate to excessive quantities. The flat topography characterizing most of the mining districts in which the deposits occur permits of a convenient arrangement of the surface plant.

Access to the deposit is obtained by two shafts which are about 150 ft. apart. One shaft is used for ventilation, drainage and exit purposes,

while the other is the working shaft. In a large mine the ventilating shaft is equipped with an exhaust fan, in the smaller mines natural ventilation is sufficient. Practically no dangerous gases are present and open lights are used.

The method of mining is a modification of top-slicing. The sublevel interval varies somewhat but approximates 15 ft. All of the coal with the exception of a portion 3 ft. in thickness, left in the roof of the room as a protection from the overburden and the caved cover is mined. The unit attacked is a block from 12 to 15 ft. square. A small crosscut 3.5 by 5½ ft. gives access to the room. The coal is excavated by pick and shoveled into small cars, which are trammed to inclines connecting the sublevels and the main or haulage levels. From 70 to 80 per cent. of the coal is won. The units are mined in sequence, the mining starting at



(Numbers show Sequence of Working Rooms)

FIG. 8.—METHOD OF WORKING A "SQUARE." FROM *Bergbaukunde*, HEISE-HERBST. VOL. i, p. 327.

the boundaries of the deposit or the edge of a panel. The line of retreat is stepped. After mining a room the props are pulled and the cover allowed to cave. Figs. 7 and 8 give details of the method.

The room is supported by three or four lines of props. A lid or head board is used on each prop and the sides of the room where necessary are supported by light lagging held in place by sprags. The props range in size from 5 to 7 in. in diameter. By means of short pieces of lagging the lid is given an area of from 3.5 to 4.5 sq. ft. From one-quarter to one-third of the area of the roof of the room is thus supported. The development drifts and crosscuts are closely timbered although the timbers are usually of small section. Main haulage ways and shafts are supported by masonry. The shaft sections are circular or elliptical. I saw but one timbered shaft of rectangular section. At a small mine close to Leipsic (the Dölitz) the working shaft was 75 m. in depth and of this the

top 15 m. was brick, the next 50 m., cast-iron tubing and the lower 10 m. brick. The shaft was 4.5 m. in internal diameter and contained two hoisting compartments and one manway. It gave access to a bed of brown coal about 40 ft. in thickness.

The general plan of development of a slice may be called a double-entry panel system. The large unit or panel is from 300 to 500 ft. square and is divided by drifts and crosscuts intersecting at right angles into smaller squares which measure from 40 to 100 ft. on a side. These squares are in turn attacked by the working drifts and the smaller units or rooms developed. The order of working off a square by rooms is shown in Fig. 8.

The inclines connecting the sublevels are served by chain lifts of a type similar to those used in the open pits. Main level haulage is either by the *kettenban* or the *seilban*. The latter is the continuous wire-rope haulage system. The cars used are of the same size and general construction as those used in the open pits.

Pumping is effected by direct-connected, electrically driven, single, two- or four-stage centrifugal pumps. Pump chambers are usually arched and supported by masonry. Significant features of the installation at one mine were the large sump space and the use of screens to prevent sand or any suspended matter from getting into the suction pipes. At one small mine visited the pumping equipment consisted of three steam pumps, one for continuous service of somewhat over 500 gal. per minute and two in reserve of a combined capacity of 1,500 gal. per minute. The head against which these pumps operated was 240 ft. At another and somewhat larger mine the pumping capacity in multistage centrifugals aggregated 3,200 gal. per minute. I was informed of a mine which was equipped with a pumping equipment of over 6,000 gal. per minute capacity. These figures will give some conception of the pumping problem. The heads against which the water must be lifted are in most cases not excessive, ranging from 100 to 250 ft.

Preparation of Coal for Market

Brown coal is graded into large lump, fist, nut and fine sizes. The first three sizes are sold without further preparation. The fine material is mixed with water and put through a common brick press. It is molded and cut into bricks about the size of the ordinary building brick. These are air dried in open sheds and sold as "nass brick." They contain about 20 per cent. moisture. The greater part of the brown-coal output is briquetted. The coal is first passed through crushing rolls which reduce it to $\frac{1}{4}$ -in. size. It is then elevated and conveyed to driers, which are constructed very much like McDougall roasting furnaces. Each of the 32 hearths is covered by a steam coil. Rabble arms distribute the coal

and transfer it from hearth to hearth. The coal enters with about 50 per cent. moisture and is discharged with about 12 per cent. It is then conveyed to bins from which it is fed to the briquetting presses. In all the plants visited there was but one type of press in use. This was driven

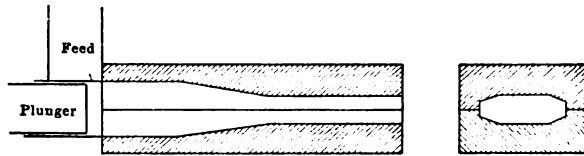


FIG. 9.—LONGITUDINAL SECTION OF BRIQUET MOLD.

by steam. The die or mold of the press is formed of two steel plates which form symmetrical halves and are firmly held in the machine. The opening between the plates narrows down from a rectangular opening of



FIG. 10.—BRIQUET TROUGH AND STORAGE SHED.

the same size as the plunger to the size or section of the finished briquet, as shown by Fig. 9. A feed spout delivers the fine coal at the large end of the die, which is steam heated, and the plunger gradually forces it through the die. The briquets are delivered apparently in a solid bar, but can be readily separated. The briquets are forced along a light

metal trough, by the impulses of the press, to the storage sheds (Fig. 10). The press makes from 60 to 75 strokes a minute. A crank and flywheel control the length of the stroke. The capacity of a press is about 130 tons per 24 hours.

The briquet is 6 to 7 in. in length, $2\frac{1}{2}$ in. wide and $1\frac{1}{2}$ in. thick, elliptical in shape. The analysis of a briquet is given in *Coal Resources of the World* as follows: Water, 14.42 per cent.; ash, 7.10; fixed carbon, 33.85; volatile carbon, 44.63. The heating value of brown-coal briquets ranges from 4,500 to 5,000 calories.

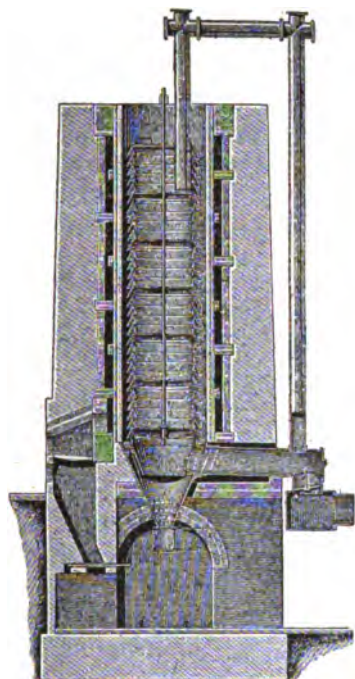


FIG. 11.—SHAFT FURNACE FOR DISTILLATION OF BROWN COAL.

In one plant visited, the Pffenershaft near Halle, part of the mine output was treated by distillation in muffle furnaces of the shaft type. The distilling chamber of the furnace is 5 ft. in diameter by 26 ft. high. The construction of the furnace is shown in Fig. 11. The brown coal is heaped over the mouth of the shaft being prevented from entering the central part of the shaft by a plate which covers the uppermost ring. The space between the walls of the shaft and the cast-iron rings, which are in the inner portion of the shaft, is occupied by the fine coal as it falls down. The gaseous products of the distillation pass between the iron rings and into the gas flues which reach the inner space. The combustible gases are used for heating the furnace. The fine coke, which

is about pea size, is discharged from the hopper. The tar is used for the production of paraffin and in the chemical industries, and the coke for heating large halls, museums, etc., being extremely effective for this purpose. The coke when used for heating, is burned in shallow open pans, burning without flame, very much like punk.

Surface Plants

Surface plants are compactly arranged and the buildings constructed of brick and steel. Steel headframes are used at the shafts. The smaller mines are equipped with a boiler plant, steam hoist, screening house, a brick plant for producing "nass brick," a power plant and bins. In the more elaborate plants crushing, drying and briquetting appliances are added. Steam power is used in most all surface operations and electrical power for underground service. In one case the mine pumps were driven by steam power. Bins are constructed of steel or steel and brick.

Production, Costs, Etc.

The production in 1912 is given as 82,339,583 and in 1913 as 87,116,343 metric tons. (*Mineral Industry*, vol. xxii.) The value of the product for 1908 is given (*Berk Hütten Kalender*, 1911, p. 178) as 67,615,200 metric tons of a value of \$45,230,000. This would give a value of \$0.665 per metric ton. From *Braunkohle*,¹ a technical magazine devoted to the brown-coal industry, the following information was taken:

Selling price of brown coal at mine (wholesale).....	\$0.72 per short ton
Estimated cost of removing overburden.....	0.06 to 0.10 per cu. yd.
Underground mining cost, estimated.....	0.45 per short ton
Cost of mining coal in open pit, not including cost of stripping (coal 25 to 60 ft. thick).....	0.09 to 0.10 per short ton.

The wholesale price of briquets ranges from \$2.38 to \$4 per ton at market centers, not at the mine. (*Bulletin 58, Bureau of Mines*, p. 14.)

Comment

There is considerable similarity between brown-coal mining methods and the methods used in mining soft iron ores such as occur on the Mesabi Range in Minnesota. The development of mining practice in two widely separated mining fields where somewhat similar physical conditions pertain is worthy of comment. The methods used upon the Mesabi are steam-shovel stripping and open-pit mining in which the steam shovel is almost exclusively used as a loading device, the milling system and top-slicing. The methods used in brown-coal mining are

¹ Serial article, "Ueber die Bemessung des Verhältnisses zwischen Kohlen- und Deckgebirgsmächtigkeit für Tagebaubetrieb im Braunkohlenbergbau," *Braunkohle*, July 20, 1909. The figures apply to the district about Halle.

stripping with the continuous-bucket excavator and open-pit mining, in which the loading is effected by the milling system, and top slicing. On the Mesabi a large proportion of boulders in the glacial drift and more or less rock makes it almost impossible to avoid the use of the steam shovel. In brown-coal mining the overburden is sand or soft sedimentaries and the mechanical excavator is more economical than the shovel. It is worthy of note that in districts in this country where similar material must be removed in stripping the scraper bucket or drag-line excavator is being used in preference to the steam shovel. It does not possess all of the advantages that are obtainable with the continuous-bucket excavator.

The greater hardness of the iron ore as compared with the brown coal and the greater cost of labor in the United States explain the use of the steam shovel for ore mining. The soft nature of the brown coal on the other hand would indicate that a more extended use of mechanical appliances would be advantageous for excavating the brown coal. In the score or more of pits visited only one mechanical cutter was observed and hand picking of the coal ruled.

Approaches to the German open-pit mines are invariably short steep inclines served by chain-haulage systems while in the Mesabi open pits the approaches are long gentle slopes, seldom exceeding a grade of 6 per cent. and more often approximating 2 or 3 per cent. The smaller outputs, the greater value of the surrounding land, the necessity for preparing the coal before it can be marketed, the greater economy in haulage and the smaller capital investment required are the probable reasons for almost general use of the steep incline in the German open pits. In the Mesabi open pits the railroad cars are taken into the pit, loaded, hauled out and made up into trains. The large outputs, the short mining season, and the flat nature of the orebodies are the principal reasons for the expensive approaches. There are cases where a skip-served incline and the double handling of the ore would be more economical than an approach of the ordinary type.

The underground methods are essentially the same in principle in both localities. The most interesting point is the comparison of the size of the top-slice unit and the time required in mining a unit. The comparison is given in the table.

Size of unit:	Mesabi	Brown Coal
Height, feet.....	13	12 to 13
Width, feet.....	14	12 to 13
Length, feet.....	50	12
Volume of unit, cubic feet.....	9,100	1,800
Tons mined per miner per shift.....	12	6 to 7
Estimated time required to mine a unit (1 shift of 2 miners per day), days.....	24	2
With 2 shifts of 2 miners per shift.....	12	1

Two important factors in top slicing are the size of the unit and the time required for the maintenance of the unit as an open chamber. It is obvious that the harder the ore the larger the unit and the longer the time that it can be maintained open with nominal support. Mesabi ore is called a soft ore. It is drilled with augers and loosened with light powder charges. The unit is timbered with two rows of drift sets spaced at 5-ft. intervals. The unit or chamber will remain open without serious failure of the supporting timbers for 30 days. This is sufficient to mine a unit of the size given. It is estimated that from 5 to 6 per cent. of the units begin to fail before they are completely mined out. Brown coal is decidedly softer than the Mesabi ore. Obviously a smaller unit and more rapid mining are necessary. The unit is approximately one-fifth the size of the Mesabi unit and is mined out in one-twelfth the time. Fewer and lighter timbers are used.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The New Electric Hoist of the North Butte Mining Co.

BY FRANKLIN MOELLER,* CLEVELAND, OHIO

TOGETHER WITH A NOTE ON PRELIMINARY CALCULATION OF FLYWHEEL MOTOR-GENERATOR SETS

BY C. D. GILPIN, CLEVELAND, OHIO

(New York Meeting, February, 1916)

THE application of electric power for driving mine hoists handling heavy loads at high speeds has recently been extended by the installation of what is probably the largest electrically driven hoist in this country at the Granite Mountain shaft of the North Butte Mining Co., Butte, Mont. While the weight of rock which is hoisted by this machine is not the largest now being handled by electric mine hoists, the speed of hoisting is so, and the combination of the high speed with the heavy load has resulted in a machine of considerable interest to all those actively engaged in mining. Aside from mere consideration of size, this installation deserves attention because of the careful attention given, not only to the question of economy of operation, but to the details of construction and installation of the entire equipment.

The conditions to be met in this installation were as follows:

Weight of rock to be raised per trip, 7 gross tons	15,680 lb.
Weight of skip	8,000 lb.
Weight of cage	1,800 lb.
Maximum depth of hoisting	4,000 ft.
Diameter of rope	1½ in.
Total weight suspended on one rope from drum	42,000 lb.
Normal hoisting speed	2,700 ft. per minute.
Maximum hoisting speed	3,000 ft. per minute.

Desired capacity—200 tons per hour from 4,000 ft. depth.

The ore is mined in several levels and hoisting is carried on from the 800-ft. level down to the 2,700-ft., which is at present the maximum depth from which the ore is raised regularly. Development is proceeding and a depth of 2,900 ft. has already been reached. The rock is handled underground in cars holding $\frac{3}{4}$ ton, and about 10 cars are the usual load for a skip. All of the ore is dumped into underground bins, which are of different capacities varying up to 250 tons. From the bins the ore is fed into the skips without passing into measuring pockets. At the surface the

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ore is delivered into a series of storage bins, from which it is discharged by gravity into railroad cars and shipped to the smelters.

With a high cost of fuel and the cost of electrical power relatively low, the most economical hoist apparatus is a simple geared hoist driven by an alternating-current motor with drum control. Its simplicity, however, carries with it certain disadvantages which become more pronounced as speeds of hoisting are increased. At this mine, where the service requires a combination of high speeds and heavy loads, the first choice of equipment is a hoist direct connected to the motor with voltage control. In the many questions that enter into the determination of the best hoist-



FIG. 1.—HOIST ROOM AT THE GRANITE SHAFT OF THE NORTH BUTTE MINING CO.

ing equipment for any given set of conditions, first cost may finally be pitted against sureness of control, freedom from interruption of service, and safety. The safety feature was given as careful attention as any of the other items entering into the consideration of the hoisting equipment to be installed.

In the present instance, a first-motion hoist with a motor-generator flywheel set was chosen as the most desirable equipment. Fig. 1 shows a part of the hoist room with the equipment installed. The hoist house is of steel, with steel sides and roof. An overhead crane and runway are provided, so that the heaviest parts of the equipment can be handled in case

repairs are necessary. The hoist is located 425 ft. from the head frame, the ropes between the head frame sheaves and the drums being supported on sheaves mounted on special towers.

The hoist consists of two 12-ft. drums fitted with a clutch, post brake and band brake. The drums are mounted on a shaft supported in three bearings, this shaft having a flanged coupling to connect to the motor. The clutches and post brakes are operated by oil cylinders, the pressure being supplied by an accumulator and an electrically operated pump. The band brakes are operated by hand wheels. All of the operating levers are grouped on a large elevated platform with double stairways. The control and reverse levers are separated, but so interlocked that when the control lever is in the "on" position the reverse lever cannot be moved. The safety devices include:

1. A mechanism for moving the control lever to the "off" position when the skip has reached a predetermined point, holding the lever in this position until the reverse lever has been moved to the opposite position, the operator being thereby prevented from starting in the wrong direction.

2. Two solenoids which automatically apply the post brake if the skip is carried too far after the current has been cut off. An indicator with large dial is provided for each drum; for accurately spotting the skip or cage the brake rings on the drums next to the middle bearing are extended 8 in., affording a large surface on which to paint marks.

On the platform in front of the operator are a panel, holding a voltmeter and an ammeter, and a target connected to the reverse lever showing which drum is hoisting. Grouped around the sides of the platform are the signals, gongs and lights.

The drums are 12 ft. in diameter by 9 ft. 4 in. face, grooved to hold 5,000 ft. of $1\frac{5}{8}$ -in. rope in two layers. The drum shells, brake rings and spiders are made of cast steel, the latter being fitted with heavy bronze bushings.

The clutches are designed to take a load of 50,000 lb. on a 12-ft. diameter, with a factor of safety on all parts of not less than 8. The clutches are of the flat friction type, consisting of two heavily ribbed annular rings faced with wood supported on a six-armed spider keyed to the shaft. These rings clamp a flat steel plate bolted to the drums, and are moved by six sets of toggle arms connected to a sliding sleeve and rock shaft operated by an oil cylinder. All of the parts of the clutches are made of steel. The clutches and motor were subjected to a load of $2\frac{1}{2}$ times their rated capacity and developed no weakness.

The post brakes are made of plates and angles in the form of a box girder. They are of the parallel acting type applied by weights and released by oil cylinders. The band brakes are for emergency service and are operated by hand wheels and screws, with provision for operation by

power later on if desired. All of the brakes are lined with basswood blocks.

The bearings are of the pedestal type with quarter-boxes adjustable both vertically and horizontally. A continuous gravity feed oiling system with tanks and filters is provided for lubrication. The drum shaft is of open-hearth forged steel, 22 in. diameter by 40 ft. $5\frac{1}{2}$ in. long, with a flanged coupling forged on end. All of the operating connections and safety devices are placed on or above the floor level, in full sight, so that any derangements of any of the working parts can be quickly observed.

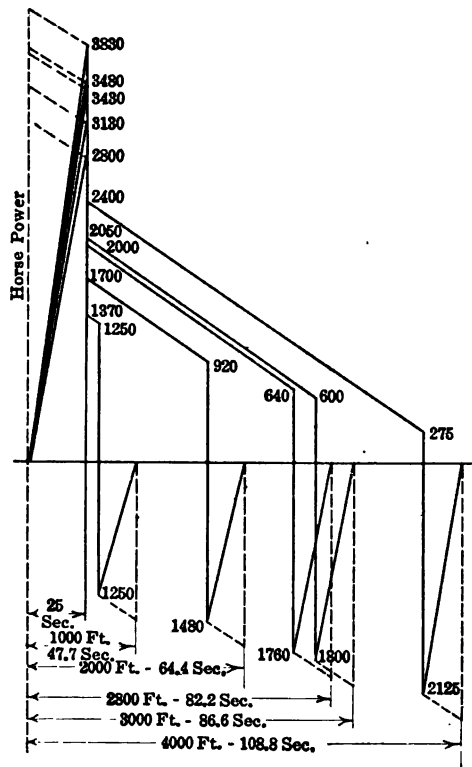


FIG. 2.—CALCULATED HOISTING SPEED AND POWER CURVES.

The combined weight of drum shaft with two drums and two clutches is 300,000 lb.; the radius of gyration is 4.86 ft.

The electrical equipment consists of a 1,850-hp., 71-r.p.m., 550-volt, direct-current, direct-connected hoist motor, directly connected electrically to a generator of equivalent capacity which is driven by a 1,400-hp., 505-r.p.m., AC motor. A 100,000-lb. flywheel is mounted between the motor and the generator of the motor-generator set. The flywheel is turned smooth and is inclosed in a steel case which reduces not only the windage friction but the noise; the latter often being a very annoying

feature with a set of this kind. Both the hoist motor and the generator are designed to carry high overloads.

Forced lubrication is used on the motor-generator set, and the wheel bearings are water-cooled. The slip regulator is also water-cooled.

The field current of the generator is not regulated directly, but by means of magnetic switches operated by the master controller. The

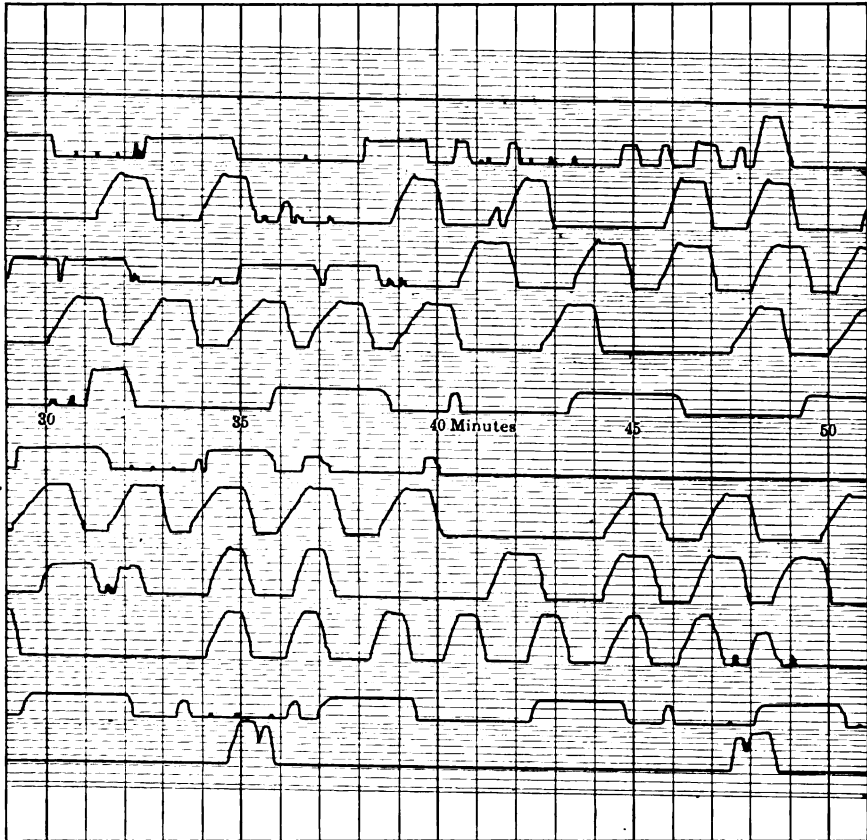


FIG. 3.—TACHOGRAPH RECORD SHOWING TYPICAL HOISTING SPEEDS AT THE GRANITE SHAFT.

wiring for the latter is all brought to a junction box on the operating platform, so that the operator may know if any changes are being made. All other wiring is carried in the basement, conduit work being used throughout with the exception of the large mains between the motor and generator which are mounted in porcelain cleats in a substantial manner. Every connection is plainly marked so that it can be checked up by reference to the wiring diagram.

The operation of the hoist is extremely easy, merely one air brake

being commonly used; this air brake, in fact, is usually not applied until the rope speed is reduced to almost nothing. At present the hoist is not pushed to its capacity and consequently the operators accelerate and retard in a leisurely manner. Fig. 3 shows some typical speed records from the tachograph with which the hoist is provided. Fig. 4 is a power curve taken when the operator was hoisting at a speed about 10 per cent. below normal from a depth of approximately 2,250 ft. Fig. 2 shows a series of calculated curves. As a matter of interest, the values from the 1,000-ft. level curve and 3,000-ft. level curve have been applied to equation 4 in the note appended to this paper, with the result that a flywheel sufficient to smooth out all the peaks is indicated, based on assumed values for demand and meter charges. Calculations made from these curves

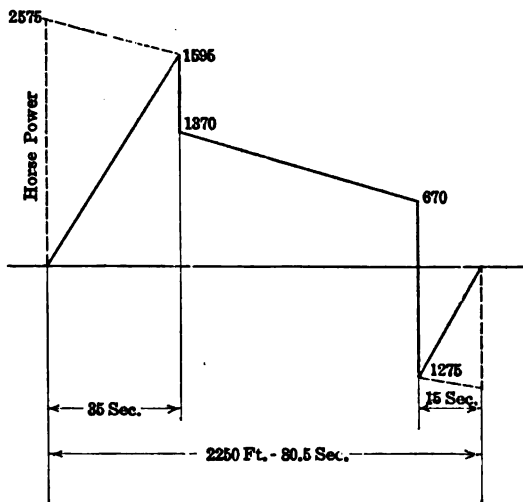


FIG. 4.—POWER CURVE WHEN HOISTING AT A SPEED 10 PER CENT. BELOW NORMAL FROM A DEPTH OF 2,250 FT.

indicate for the 1,000-ft. level that a 90,000-lb. wheel will be required, while for the 3,000-ft. level a 111,000-lb. wheel will be necessary, a minimum time of 15 sec. being allowed for loading in each case. These figures are approximate and are in no way related to those prepared by the manufacturers of the electrical equipment, who, of course, have access to more complete data on the characteristics of the electrical equipment. In these calculations, the slip was taken at 20 per cent.

The input curve at present reaches a value of 1,140 kw. and consists of a series of peaks, since the interval between trips is too great to give the constant power input that would be attained under maximum rates of operation. The switchboard instrument indicates a no-load loss for the flywheel and motor shunt field, etc., of approximately 90 kw. or about 120 hp.

It should be borne in mind that the equipment was installed for conditions which will prevail in the future, as much as for those which now are attained.

Preliminary Calculation of Flywheel Motor-Generator Sets

The subject of motor-generator hoisting sets has been covered so many times that it is difficult to offer anything new. The writer has endeavored, however, to develop a few general principles which may be of use to those interested in this subject.

In the first place, this system is often considered for hoists where it has no proper application. Take for instance a hoist where the rope speed is comparatively low. Though the peak loads on such a hoist may indicate a saving in demand charge which appears attractive, it should be borne in mind that the efficiency of an Ilgner system is much lower than that of a straight induction motor, so that the demand charge will not be

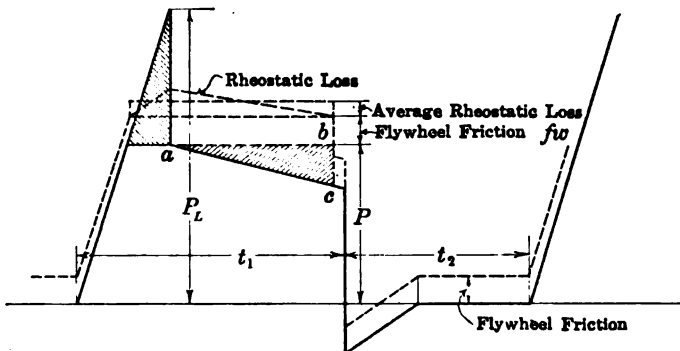


FIG. 5.—POWER CURVE USED IN CALCULATION OF FLYWHEELS FOR MOTOR-GENERATOR SETS.

reduced as much as would appear. Moreover, if the hoisting speeds do not require a motor-generator type of control, the substitution of an Ilgner system for an induction-motor drive will greatly raise the fixed charges, which will generally more than offset the saving in demand charges.

There is another class of hoists in which the splendid control of the motor-generator system is essential, but in which the peaks are not of sufficient magnitude to indicate the use of a flywheel. In cases of this kind, if the company supplying power will disregard peaks under 5 seconds, it does not pay to add the flywheel, since the acceleration load area is triangular and much of it will not carry any charge. As the load comes on gradually, there is little disturbance on the power lines, so that the power company can afford to make such a concession.

Finally, there are many instances in which a short inspection of the load cycle and the power contract will make it certain that a flywheel system is necessary. Just how much of the peak it is desirable to cut off is

not so easily determined. The following method of calculation may, perhaps, be of interest.

A curve should be plotted, similar to the one shown in solid lines on Fig. 5, this curve to represent the power at the flywheel shaft. The peak cost per horsepower per trip (C_1) should next be determined by dividing the peak charge per horsepower per month by the probable number of trips per month, allowing for the losses in the alternating-current motor. The current charge (C_2) per horsepower-second should be figured.

It is obvious that the acceleration peak of the curve will be comparatively easy to do away with, and this point should be first considered. Assuming a constant maximum slip, s , for the flywheel (expressed as a decimal fraction of normal speed), and calling the weight of the flywheel, W , energy from the wheel (in horsepower-seconds) = $\frac{Wv^2(2s - s^2)}{2g \times 550}$ where v is the equivalent linear velocity of the flywheel in feet per second ($v = 2\pi$ rev. per sec. $\times$ radius of gyration of wheel).

This expression may be more conveniently written, nW ; and to obliterate the acceleration peak it is only necessary to equate this value to the energy in the acceleration peak above the gravity load.

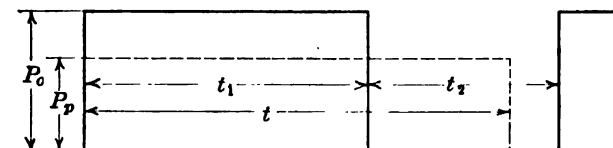


FIG. 6.—LOAD CURVE.

Referring to Fig. 5, re-acceleration would be re-represented by the area, *abc*. The actual peak at the wheel is P plus the wheel friction and the rheostatic losses, and the power curve has assumed an approximately rectangular shape. It would not, therefore, be far out of the way in figuring the desirability of a further increase in the flywheel, to assume that the load curve is a rectangle as shown in Fig. 6, in which t_1 is the time of hoisting up to retardation, P_o the average power at the wheel required by the hoist during this period, and t_2 the time of retardation plus the time of rest. P_p is the power delivered at the wheel shaft by the alternating-current motor, the difference between P_o and P_p being supplied by the wheel. The energy $P_o t_1$ then equals $P_p t$. The negative power has been disregarded, as it does not ordinarily enter largely into the final result.

Let f equal friction horsepower per pound of flywheel, and T = total time wheel runs per trip. Since approximately all the power passes through the motor-generator set while the latter is accelerating or retarding, the average rheostatic loss in power equals $\frac{s}{2} (P_p + fW)$. The peak

load, therefore, equals $P_p + fW + \frac{s}{2}P_p + \frac{s}{2}fW = \left(1 + \frac{s}{2}\right)(P_p + fW)$

$$P_p = P_o \text{ less the power supplied by the wheel} = P_o - \frac{nW}{t_1}$$

Substituting,

$$\text{peak load} = \left(1 + \frac{s}{2}\right) \left(P_o - \frac{nW}{t_1} + fW\right)$$

Let the peak load cost of power per trip = F_1 .

$$F_1 = C_1 \left(1 + \frac{s}{2}\right) \left(P_o - \frac{nW}{t_1} + fW\right)$$

The rate of change of peak cost with regard to wheel weight is the first derivative of F_1 , or $\frac{dF_1}{dW}$

$$\frac{dF_1}{dW} = C_1 \left(1 + \frac{s}{2}\right) \left(f - \frac{n}{t_1}\right) \quad (1)$$

With ordinary values for n , t_1 and f , this rate of change will be negative, showing a decrease in peak cost for an increase in wheel weight.

The energy consumption at the flywheel shaft is the sum of three quantities: the energy demanded by the hoist, the energy consumed by the flywheel, and the energy lost in the slip regulator rheostat. The first of these quantities is represented by $P_o t_1$, and the second by TfW , it being remembered that T is the total running time of the wheel per trip of the hoist.

The third quantity (E_s) is equal to the energy lost when introducing resistance plus the energy lost while resistance is being cut out; it may be expressed as follows:

$$E_s = \frac{s}{2} (P_p + fW)t_1 + \frac{s}{2} (P_p + fW)(t - t_1) = \frac{s}{2} (P_p t + tfW) = \frac{s}{2} (P_o t_1 + tfW)$$

The quantity t is a variable dependent on W , and may be expressed in terms of the latter; but for approximate calculations of this kind, it is much simpler to equate t to T , which is its limit. The error so introduced, as will be shown hereafter, will be small, when determining the rate of change of energy cost.

The total energy, E , may therefore be approximated as follows:

$$E = P_o t_1 + TfW + \frac{s}{2} (P_o t_1 + TfW)$$

Let the meter cost per trip be F_2 . Then,

$$F_2 = C_2 \left[P_o t_1 + TfW + \frac{s}{2} (P_o t_1 + TfW) \right]$$

The rate of change of meter cost = $\frac{dF_2}{dW}$

$$\frac{dF_2}{dW} = C_2 \left(Tf + \frac{s}{2} TF \right) = C_2 \left(1 + \frac{s}{2} \right) Tf \quad (2)$$

(It will easily be seen that if t equals 0 instead of T , the term $\frac{s}{2}$ will be eliminated; with 15 per cent. slip, $\frac{s}{2} = 0.075$. The error is therefore usually much less than $7\frac{1}{2}$ per cent.)

$\frac{dF_2}{dW}$ is a positive expression and shows F_2 increasing with the weight of the wheel. If the derivative of F_1 is negative and is greater numerically than the derivative of F_2 , the power cost per trip will decrease continuously as the wheel weight is increased, until the peaks are entirely smoothed out. If the derivative of F_2 is the greater, any increase in the wheel weight will increase the cost of power. This relation may be expressed as a ratio of the two derivatives

$$\frac{dF_1}{dW} \div \frac{dF_2}{dW} = \frac{dF_1}{dF_2} = \frac{C_1 \left(1 + \frac{s}{2} \right) \left(f - \frac{n}{t_1} \right)}{C_2 Tf \left(1 + \frac{s}{2} \right)} = \frac{C_1 \left(f - \frac{n}{t_1} \right)}{C_2 Tf} \quad (3)$$

If e_1 equals the average efficiency of the alternating-current motor; R_1 , the peak charge per horsepower per month; R_2 , the meter charge per horsepower-hour; D , the working hours per month, and Z , the average trips per working hour, then

$$C_1 = \frac{R_1}{DZe_1}; \quad C_2 = \frac{R_2}{3,600e_1}; \quad T = \frac{3,600}{Z};$$

and

$$\frac{C_1}{C_2 T} = \frac{R_1 3,600 e_1 Z}{D Z e_1 R_2 \times 3,600} = \frac{R_1}{D R_2}$$

Substituting in (3),

$$\frac{dF_1}{dF_2} = \frac{R_1}{R_2 D f} \left(f - \frac{n}{t_1} \right) \quad (4)$$

If the numerical value of $\frac{dF_1}{dF_2}$ exceeds unity and is negative, all the peaks should be removed; otherwise additional flywheel weight will only increase the cost of power.

It may be objected that a flywheel merely large enough to obliterate the acceleration peak will not in every case produce an approximately rectangular-shaped power curve. This objection is met by an inspection of the typical load cycles shown in Fig. 7. In both cases h will be the average power for the time $t_1 + t_2$. Let W_1 be the wheel required to reduce the triangular load to h , and W_2 be a similar wheel for the rec-

tangular load. Without going into the mathematics on this point, the relation of the wheel weights is as follows:

$$\frac{W_1}{W_2} = \frac{(t_1 + 2t_2)^2}{4t_2(t_1 + t_2)}$$

When t_2 is very large as compared to t_1 , W_2 will be as large as W_1 ; and since the reduction in peak for the rectangular curve is much less than for the other, if it is advantageous to reduce the rectangular peak, it will be still more so to smooth out the triangular curve. The other extreme is represented by the value of t_1 which is large compared to t_2 ; in actual practice, this ratio will hardly exceed 10 to 1 in cases where a flywheel is applicable. $\frac{W_1}{W_2}$ then becomes equal to $\frac{1.44 t_1^2}{0.44 t_1^2}$ or to 3.3, and $W_1 = 3.3W_2$.

The rectangular curve will show a reduction in peak of less than one-tenth, while the triangular curve will be reduced approximately one-half.

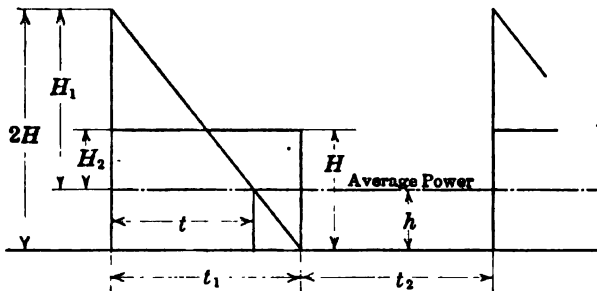


FIG. 7.—TYPICAL LOAD CYCLES.

Since the losses are roughly proportional to the wheel weights, the triangular load will show a ratio of saving over that of the rectangular load in the neighborhood of 5 to 3.3. It would appear, therefore, that the rectangular form of curve, for any ordinary conditions, is the limit of the other forms. Therefore, it may be assumed that if any hoisting cycle shows by means of equation (4) that its input should be made constant, a saving in power bills will result by so doing. The converse is not necessarily true; *i.e.*, a curve of markedly triangular form, which by the use of equation (4) shows a value of somewhat less than unity for $\frac{dF_1}{dF_2}$, should be further investigated by means of one or two trial wheel weights.

Referring again to the formula for $\frac{dF_1}{dF_2}$, approximate values may be placed on all the constants. The expression $\frac{R_1}{R_2}$ would probably lie between the ratios $\frac{50}{1}$ and $\frac{400}{1}$ with various power contracts; D (working hours

per month) should approximate 600; f may be estimated roughly as 0.001 for ordinary speeds of flywheel (1 hp. per 1,000 lb. of wheel). Quantity n may be taken as 0.4, and the value of 0.15 for s would not be far off.

Substituting the greatest ratio of $\frac{R_1}{R_2}$ and the other values as stated,

$$\frac{dF_1}{dF_2} = \frac{400}{600 \times 0.001} \left(0.001 - \frac{0.4}{t_1} \right) = 666.7 \left(0.001 - \frac{0.4}{t_1} \right) =$$

$$0.6667 - \frac{266.7}{t_1}.$$

Equating $\frac{dF_1}{dF_2}$ to -1 , which is the ratio at which it will cease to be profitable to add flywheel, t_1 will equal 160 sec.

Substituting the minimum ratio of R_1 and R_2 $\left(\frac{50}{1}\right)$, t_1 becomes 31 sec.

Therefore, when the time of hoisting, less the time of retardation, is less than 31 sec., and the working hours per month are about 600, it is extremely likely that a sufficient flywheel should be supplied to make the power input constant; if it is more than 160 sec., no flywheel should be added over that necessary to remove the acceleration peak. If 20 per cent. slip were assumed instead of 15 per cent., n would have a value of about 0.5, and the preceding time values would be increased about 25 per cent.

No attempt has been made to discuss the question as to the propriety of treating the slip, s , as a constant, but the writer believes that for most cases this value should be as great as the design of the motor-generator set will allow. Values of n can readily be worked out for various speeds of flywheels, but a discussion of the probable values of f , by those engaged in the design of this type of apparatus, would undoubtedly be of interest.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Application of Electric Power to Mining Work in the Witwatersrand Area, South Africa

BY J. NORMAN BULKLEY, S. B., NEW YORK, N. Y.

(New York Meeting, February, 1916)

As electrical power is used to a greater extent on the Rand than in any other mining center, it is thought that a short description of the methods used and results obtained may be of interest. In comparing Rand practice with that on other fields, several general factors should be borne in mind: First, the mines are all working on large tonnages of low-grade ore (average \$6.50). The annual crushing rate is about 27,000,000 tons. Second, the mines are all grouped under the financial and engineering control of mining houses, so that the results of a group, rather than an individual mine, are open for study, and practice follows more uniform lines. Third, coal of fair quality (about 12,000 B.t.u.) can be obtained for \$2 to \$3 per ton. Good supplies of condensing water are, however, scarce, and have to be carefully stored.

Power Supply

Power is supplied by the Victoria Falls & Transvaal Power Co., Ltd., and its subsidiary, the Rand Mines Power Supply Co. Both companies, which are practically one from an engineering point of view, operate the following stations, all feeding into the same system.

Station	Capacity Installed Turbo Alternators	Capacity Installed Steam-Tur- bine Driven Centrifugal Compressors
Brakpan.....	Two 12,500-kw. sets Two 3,000-kw. sets	
Simmer Pan.....	Six 3,000-kw. sets	
Rosherville.....	Two 11,000-kw. sets Five 9,600-kw. sets	Six 3,500-kw. machines Three 7,000-kw. machines
Vereeniging.....	Two 9,600-kw. sets Two 12,000-kw. sets	
	162,200-kw. sets	
Total steam-turbine capacity installed, 204,200 kw.		

There are also at the Robinson Central air station six motor-driven centrifugal compressors, each of 3,500 kw., equal to 22,000 cu. ft. per minute at 120 lb. capacity at 3,000 r.p.m. Power is supplied from these stations as 50-cycle, three-phase current generally at 40,000 volts,

though the western end of the line is at present worked at 20,000 volts, and the Vereeniging tie line at 80,000 volts. The supply to the consumer is generally at 2,100 and 525 volts. The combined output of these stations is about 2,000,000 kw.-hr. daily, exclusive of air, with a peak load also exclusive of air of about 92,000 kw.

In addition to the supply furnished by the above companies, three groups furnish their own power, viz.:

	Station, Kilowatts
Randfontein Group.....	26,000
East Rand Proprietary Co.....	20,550
Kleinfontein Group.....	6,000

Price of Power

Standard contracts of the Victoria Falls & Transvaal Power Co. are for a period of not less than 12 years at 0.525 d. per kilowatt-hour as long as the monthly load factor is above 0.70, the load factor being based on the hour of maximum consumption. Provision is made for a periodical revision depending on cost of production and a division of profit with the consumers after certain deductions are made. As the other stations do not supply consumers outside their own group, no figures for them can be given.

APPLICATION OF ELECTRIC POWER

Reduction Works

(A) *Stamp Mills*.—In the modern mills of 2,000 to 1,900 lb. falling weight, the stamps are arranged five stamps on a camshaft, two camshafts being driven from a countershaft driven from one 50-hp. squirrel-cage motor running at from 500 to 600 r.p.m. This arrangement is generally the most convenient as it permits of overhanging both line and countershaft pulleys, so that belts can be easily removed, and the placing of motors on foundations at ground level, doing away with all motor platforms. It also makes a convenient arrangement when it is necessary to hang up stamps. When squirrel-cage motors are used, the absence of all brush gear permits the starting gear to be placed in front of the mortar box so that the mill man has a good view of the battery in starting up. With squirrel-cage motors there is no difficulty in starting up with stamps all down after a failure of power supply. Starting up can also be considerably eased by slacking off the belt tighteners. In some cases of conversion of old mills from steam to electric drive it was found impossible, owing to structural details, to adopt the plan outlined above; therefore a comparison of power required for such cases may be of interest.

Mill A is of 75 1,250-lb. stamps all driven from one motor through line shaft; line shaft was in good alignment.

Mill B is of 120 1,250-lb. stamps weighted up to about 1,500 lb. falling weight and has two line shafts each with a motor driving 60 stamps; conditions of line shaft not so good as in Mill A.

Mill C is of 100 1,900-lb. stamps of which usually only 50 are running; drive is 10-stamp arrangement described above. Results are given in following table:

Mill	Monthly Tonnage	Kw.-Hr. per Ton Crushed	Number of Motors
A	14,910	10.42	1
B	28,260	8.13	2
C	33,454	4.84	10

(B) *Tube Mills*.—These are usually driven by belts direct from a 500 to 600 r.p.m. motor to tube-mill countershaft. Direct coupling to pinion shaft though flexible couplings with motor speed of 250 r.p.m. has been used in a few cases but without much success as the jar from the mill tends to break down the motor windings. On account of the high starting torque required, motors must be of the slip-ring type, and if the motor is properly proportioned to the mill with heavy starting resistances, there is no necessity for the employment of clutches or belt-shifting devices. The usual motor is 125 hp. operating at 600 r.p.m. for a 5 ft. 6 in. by 22-ft. tube mill; or 100 hp. for a 5 ft. 9 in. by 16-ft. mill.

(C) *Other Motors*.—The motors required for rock breakers, conveying plant and cyanide works are generally standard squirrel-cage motors belted directly to individual machines.

In the cyanide works a number of centrifugal pumps with direct-coupled motors were used, but owing to the necessity of adjusting speeds to suit heads, these have been discarded in favor of belt drive.

The number of motors required for a modern reduction works of about 40,000 tons monthly capacity will be roughly 125, and by intelligent selection the number of sizes required can be reduced to six or seven, so that spares will be a minimum.

The load factor of reduction works with 20.7 tons per stamp duty can easily be kept about 86 per cent. and the average distribution for the newer works (40,000 tons per month capacity) will be as follows:

	Kw.-Hr. per Ton Milled
Stamp mill.....	4.29
Tube mill.....	6.00
Tailings wheels.....	0.87
Cyanide works.....	1.55
Breaking and sorting.....	0.68
Mechanical haulage.....	0.20
Lighting.....	0.23
Mill water supply pumps.....	0.65
Total for reduction works.....	14.45

PUMPING

Fortunately the majority of the Witwatersrand mines are comparatively dry, 10,000 gal. per hour units in duplicate usually being ample to handle all the water. In some sections, however, there are heavy flows, the East Rand Proprietary mines requiring pumps of 60,000 gal. per hour capacity. Practically all of the pumping plant is electrically driven, there being left in service few if any pumps of the Cornish type. Up to recent years all of the pumps have been of the triplex single-acting plunger type about equally divided between the horizontal and vertical patterns. Recently, however, speeds have been increased up to 60 r.p.m. and more horizontal pumps are used. Heads run from 1,000 to 1,700 ft. Most of these pumps are belted, not geared, to the motors, as experience has shown that the belt drive has lower maintenance cost. The depth of the shafts and amounts of power required makes the cost of shaft cables prohibitive at 500 volts, so high-tension cables are usually taken down the shafts. As high-tension motors, particularly those of moderate size, are unsuited for use underground, step-down transformers are required. Generally a three-phase transformer with oil switch on high-tension side is provided for each motor so no secondary switches are required. These transformers with their switches, etc., are placed in a fenced-off chamber to prevent any danger from the high-tension circuits, only the switch-operating handle and secondary circuits coming outside. A typical arrangement of such a pump station is shown on Fig. 1. The high-tension wiring in the transformer station is all bare copper carried on standard high-tension line insulators, the whole, together with necessary current transformers, switches, etc., being carried on an angle-iron framework so that all wiring can be fitted above ground. This particular station was laid out for 6,600 volts working and motors operated at 110 volts.

Centrifugal Pumps.—The advent of the high-lift centrifugal pump, notwithstanding its lower efficiency, was particularly attractive to engineers on account of its lower first cost and smaller space requirements. Several centrifugal pumping plants were therefore installed. These early plants, however, proved a source of trouble as the necessity of clearing the water of even traces of grit was not fully realized and the settling sumps provided were too small for their work. The experience gained showed what had to be done. The Durban Roodepoort Deep Gold Mining Co. installed a centrifugal pumping plant consisting of two units each handling 375 Imperial gallons per minute against a total head of 2,490 ft. For clarifying the water, rim launders were used to take off the water from the settling to the suction sumps. Each of these pumps is fitted with a 550-hp., 1,500-r.p.m. motor. The East Rand Proprietary Mines also have in service eight

centrifugal pumps of 1,000 Imperial gallons per minute against 1,150 ft. head each equipped with a 550-hp., 1,500-r.p.m. motor. So far as the writer knows, both these plants are proving satisfactory.

AIR COMPRESSING

Air for the central mines of the Rand Mines is supplied by the power company from steam-driven compressors at Rosherville and motor-driven machines at the Robinson Central station. These stations supply about 2,250,000 tons of compressed air per annum. The machines are fully described in other papers.

The power company does not supply air to any other than the Rand Mines group. Owing to the bad influence of the compressor load on the load factor when the mine is operated, as is usually done on single-shift, the engineers of the remaining group have preferred, where steam plant was not entirely discarded, to keep the compressor on steam. The East Rand group operates a steam-driven, central air plant with reciprocating compressors and the Randfontein group operates a similar plant but with motor-driven units supplied with power from its own central station. There are a number of motor-driven compressors in use where there is no steam plant available. All these are of the reciprocating type, as the size of the largest unit, 7,500 cu. ft. per minute free air to 100 lb. does not allow the construction of an efficient turbine machine, the limit of which is about 10,000 cu. ft. free air. Nearly all are direct connected to either synchronous or induction motors, very few being belt driven. Two general types are in use, the vertical high speed and the horizontal low speed, both being compounded. Practically all of the later machines are fitted with some form of plate air valves automatically operated, as these have been found to give better results in every respect than any of the mechanically operated valves.

Governing.—Up to the present no satisfactory method of governing by means of a variable-speed motor has been developed; so resort must be had to some mechanical means with a constant-speed motor. The two methods commonly used are: (1) Throttling the intake; (2) opening an auxiliary governing valve for a portion of the stroke.

The first method is, of course, very simple, but throws the entire load on and off the compressor as the throttle opens and closes. Also, the efficiency is not as good as with the second method.

With the second method, by properly arranging the valve gear on the auxiliary valve, the compressor may be run at any desired fraction of the capacity with fairly good efficiencies on light load. Both methods work very well in practice.

Air Pressure.—The supply from the stations is at 120 lb. at the station and is calculated to be 100 lb. at the mine column. The usual

pressure of the individual mine plants is 80 lb. but there is a decided tendency toward higher pressures.

Meters.—The use of purchased air necessitated the installation of air meters. With the accurate records of air consumed study showed startling variations in the amount of air used per rock-drill shift on the various mines, and further study led to the development of systems for the regular inspection and repair of all drills, inspection of pipe lines and air pressures underground, all of which when summed up had the effect of reducing the air consumption and drilling costs by large amounts.

HOISTING

There are in operation 143 electrically driven hoisting engines, exclusive of winches. The combined continuous rating of these hoists amounts to over 74,000 hp., with an average of 517 hp. Hoisting engines may conveniently be considered under three classes: (a) winches; (b) sinking engines; (c) main hoists.

Winches

These are usually small engines with drums 42 in. diameter by 30 in. face, and a duty rarely exceeding 4,000 lb. rope pull at 500 ft. per minute. They are generally of the simplest construction and operated by a standard slip-ring motor through double-reduction gearing. Control is by means of a tramway-type reversing controller with metal resistances the whole being mounted on the hoist frame so as to be readily portable. Such winches are rapidly displacing the air winches where power is available underground. In many cases, however, owing to the temporary nature of their use it does not pay to install special cables, and as air is required for drilling, the air winches are continued in service in spite of their inefficiency. In one case which came under the writer's notice, the question of increasing the compressor capacity arose. Investigation of the use of air showed a double 7 in. by 10-in. air winch in service. This was replaced with a motor-driven winch costing not over \$1.50 per month for power, with the result that it was possible to put six additional drills on the compressor, and no increase in plant was required.

Sinking Engines

The high speed and great size of the main engines required on the deep shafts of the Rand precludes their being used during the sinking period and it is customary to install for this work special temporary engines which are usually placed in front of the positions to be occupied later by the main engines.

Sinking engines must be capable not only of handling the buckets or skips but also of swinging all shaft timbers, etc., into place easily and safely without the necessity of reslinging them on chain blocks or tackles. Owing to difficulties in control, this requirement is not easily met with steam winders, though it presents no difficulties with electric drive. Furthermore, as the work of these hoists finishes with the sinking period, capital expenditure must be kept as low as possible. Consideration of these points has led to the development of the following type of sinking engine which has proved extremely satisfactory in service.

Engines are double drums 8 ft. diameter by 2 ft. 3 in. face. Each drum with its brake ring is keyed fast to an independent shaft with two bearings. These shafts also have keyed to them a main gear for each drum. The pinion shaft is common to both drums and carries the two loose pinions, which are clutched to the shaft by sliding-jaw hand-operated clutches. The brakes are of the post type, hand-operated through rack and pinion gear with hand wheel. A foot-controlled band operating on a brake disc on pinion shaft is also provided for maneuvering. The motor is of standard two-bearing slip-ring induction type of about 500 r.p.m., connected to the pinion shaft through machine-cut herringbone gears and flexible coupling. Control is by means of a liquid rheostat with pump and movable weir, reversing being accomplished by means of either contactors or oil switches. The usual load on these engines is 3,000 lb. net rock at a hoisting speed of 1,000 ft. per minute, though in laying out a new sinking plant the writer would be inclined to raise this speed to say 1,500 ft. The plant for the 3,000 ft. Central Shaft of the Cinderella Consolidated Gold Mining Co. was laid out before satisfactory delayed-action electric blasting fuse was available. Since this was the first deep shaft in the Transvaal to be sunk with electric sinking engines, it was deemed advisable to provide a sure means of hoisting the sinkers away from the blast after the round had been lit, and thus prevent accidents which might arise through failure of power supply.

This was accomplished by providing a synchronous motor fitted with necessary starting motor, an exciter wound for double normal voltage and an 8-ton 10-ft. diameter flywheel running at 750 r.p.m. After the blasting signal had been given, this set was started up in the ordinary way and connected to the line through a reverse power relay; the sinkers were not allowed to light up until they had received a signal that this had been done. Then in case of power failure the reverse power relay operated, leaving the synchronous machine driven by the flywheel operating as generator disconnected from the line but connected to the hoist so that the bucket could be pulled away. Actually, however, as it was necessary to stop the set after the blast, the reverse relay was tripped by hand every blast, and the sinkers hoisted by means of the flywheel. The records show that through power failure this set was required three times

in two years' work. The operation of the entire plant was most satisfactory and the control was so effective that the general feeling of the underground staff was strongly in favor of electrical operation as against steam.

Sinking Incline Shafts

With these shafts the conditions differ, for sinking is generally carried on through a considerable portion of the life of the mine and the whole of the available space in the shaft above the lower levels is usually required to carry on the necessary work of the mine, so that it is hardly possible to provide a special sinking engine on the surface. If the main

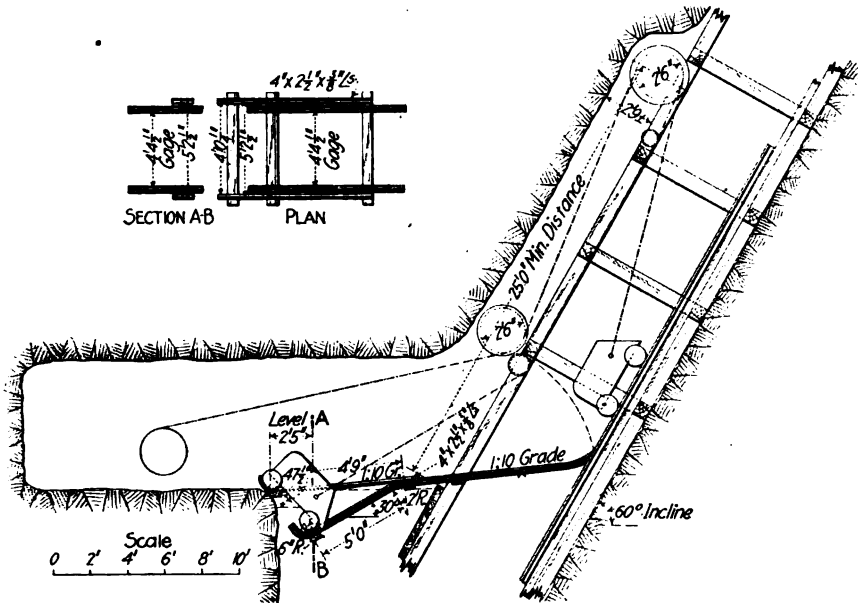


FIG. 2.—1½-TON SINKING SHIP AND OUTLINE ARRANGEMENT OF TIP, KIMBERLEY SHAFT, ROODEPOORT UNITED MAIN REEF GOLD MINING COMPANY, LTD.

engines are used, the big skips are apt to give trouble on the temporary tracks at the shaft bottom as well as being difficult to load by shovelling. It is also usual to have only one skipway carried down to the bottom, necessitating unbalanced hoisting. Where electric hoists are used, unbalanced hoisting is troublesome because the motor is usually proportioned to give its best economy on balanced load and will overheat on continued use with unbalanced full load requiring reduced skip loads. All these considerations require the use of some special arrangement for sinking incline shafts. One of the most common schemes is the use of a small winch installed in the bottom level crosscut and hauling to the main skip box, the whole arrangement being moved down when the station of

the next level is reached. The frequent moving requires that the winch must be self-contained, portable, and that the headsheaves, tip, etc., may be cheaply erected and with a minimum amount of excavation that will not be utilized in the permanent station. Such an arrangement is illustrated in Fig. 2. With this scheme the only extra excavation is the V-shaped cut, about 20 ft. long by 2 ft. 9 in. high, above the center of the skip way, to clear the sheaves and rope. The tipping gear consists of a pair of ordinary drop rails and four pieces of bent rail, together with some timber, so that the whole is cheap and readily portable. This arrangement needs no further description; it has proved extremely satisfactory in service.

Main Hoists

The enormous capacity of the main power supply stations permits extremely large amounts of power to be thrown on and off without affecting the stations. Consequently it is unnecessary to consider the installation of any flywheel sets to equalize the power demands on the line, so that the choice of rheostatic or Ward-Leonard systems of control is uninfluenced by any considerations in regard to the stations, and the choice can be made entirely on the conditions governing the particular mine. Under these circumstances the choice of system would depend largely upon the required speed of hoisting and whether hoisting when once started, can be carried on steadily without intermission. The question of capital cost must also be considered as the Ward-Leonard system must always be higher in first cost.

In regard to speeds, rheostatic control is unsafe for speeds beyond those which can safely be controlled (even when negative torque is exerted) by means of mechanical brakes. Counter-current braking cannot be relied on as alternating-current readings are not definite indications of torque. This speed limit for mechanical braking, the writer places at about 1,500 ft. per minute. It is true that in an attempt to overcome this difficulty several high-speed rheostatic control plants have been installed with eddy-current brakes, and these plants have worked fairly well, but the addition of the necessary extra parts, including storage-battery motor generator, etc., for operating the brakes, makes the cost nearly or as high as for a Ward-Leonard set and loses some of the incidental advantages of the latter. However, where the required capacity can be obtained with speeds not exceeding the limit set above, rheostatic control will give extremely satisfactory service. As the low-speed alternating-current motor required for direct coupling to the drum shaft is necessarily a machine of large diameter with little length along the magnetic iron, the frame is generally weak mechanically. This in connection with the poor efficiency and power factor of these slow-speed machines makes direct coupling undesirable except for

machines of the largest size; therefore the majority of rheostatic hoists are single-reduction geared with helical machine-cut gears. On the Rand, where it was necessary to convert a number of existing steam hoists to electric drive, the gear-shaft bearings were placed on the motor base so as to make a self-contained unit and the gear shaft coupled to the existing crank shaft through a flexible coupling, an arrangement which gave minimum change.

Hoist Control.—Control for rheostatic hoists was by means of liquid rheostat with movable weir for torque control and oil switches or air-brake contactors for direction control, both of these controls being operated from one lever. The oil switches on the smaller hoists were mechanically operated. On the larger machines compressed air and solenoids were used. However, experience proved the use of compressed-air operation unsatisfactory. Contactors were, of course, solenoid operated.

Ward-Leonard Hoists.—While the addition of the motor-generator set and the necessary control wiring make this system seem very complicated in written description, in reality after it is once installed it gives no more trouble than rheostatic control. As it is possible to use dynamic electrical braking with direct-current motors no speed limits are imposed by considerations of safety in the hoist itself, speed being governed by the permissible hoisting speed in the shaft. In this case the mechanical brakes are required for little more than locking devices at the end of the trip. This was clearly brought out at the Cinderella Consolidated shaft (4,040 ft. in depth) where after two years' service the tool marks were not worn off the brake path, the mechanical brakes not having been applied until the last 6 in. of drum travel. Owing to the high speeds usually employed with Leonard hoists, the motors are of fairly large size so that a good motor for direct coupling to the drum shaft can be obtained and thus avoid commutator troubles which might arise through pounding of the gears. The hoist motors are provided with interpoles and sometimes compensating pole face windings, with a result that sparkless commutation is secured and commutator troubles are practically unknown. Speeds of the motor-generators are only limited by considerations of mechanical and electrical design and can be fairly high, thus insuring an efficient machine at reasonable cost. These sets are usually provided with both interpoles and compensating windings on the generator. In some cases the Deri distributed shunt winding is used with marked success. The writer has installed motor-generator sets for hoist service that after three years of continuous service showed polished commutators which had never been turned down and which were in better condition than on the day they were started.

Control.—This is by means of the well-known regulation of the generator field and hardly needs comment. There are systems of control using rotary converters, and others with what might be termed, "buck-

ing motors" but so far as the writer knows, these have not been used on the Rand. When considering Ward-Leonard control, one fact should be borne in mind and that is that every position of the control lever is equivalent to fixed generator excitation and as the hoist motor has shunt characteristics the hoist speed must be practically definite, no matter what the load is or even if the torque be negative. This, of course, makes the control very easy for the driver.

RESULTS WITH ELECTRIC WINDING

In a paper of this sort it is practically impossible to present tabulated comparisons of costs with steam and electric winding, as in most cases the steam winders were supplied from boiler plants in common with other

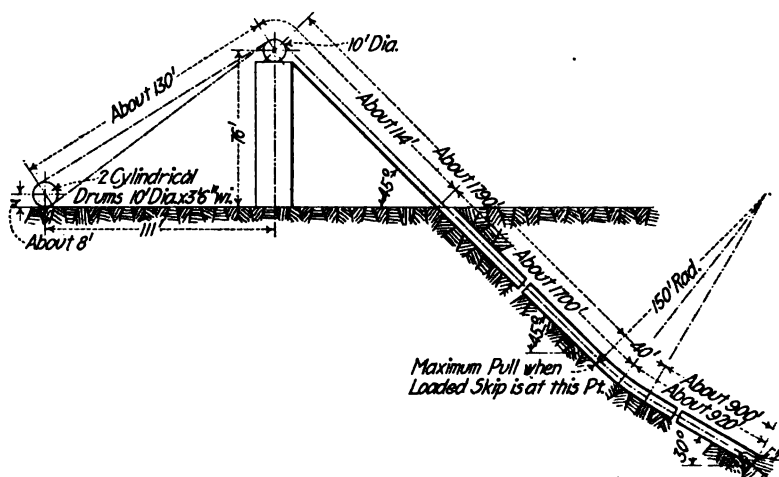


FIG. 3.—RHEOSTATIC WINDER 5-TON HOIST NO. 2 SHAFT, VAN RYN GOLD MINING ESTATE, LTD.

engines so the amount and cost of steam used by the winders was a matter of doubt. Careful comparisons showed the costs to be in favor of electric winding, the saving in the case of some of the deep shafts (4,000 ft.) amounting to at least $12\frac{1}{2}$ c. per ton. One unlooked for result was a marked decrease in the cost of maintenance of shaft guides, owing to the steadier turning moment of the electric winder. As to safety and reliability there can be no question but that under Rand conditions the advantage is strongly with the electric hoist. As a comparison of the efficiencies of Ward-Leonard and rheostatic control two shafts having conditions as nearly alike as possible have been selected. These are No. 2 of the Van Ryn Gold Mining Co. (shown in Fig. 3) and that of the Meyer & Charlton Gold Mining Co. (shown in Fig. 4). The Van Ryn is equipped with a geared three-phase rheostatic winder

handling 5-ton rock loads at 1,500 ft. per minute, with a monthly tonnage of about 22,000 tons. The Meyer & Charlton Gold Mining Co. has a direct-coupled Ward-Leonard hoist operating at 2,500 ft. per minute, with 5-ton skips of design and weight similar to those at the Van Ryn, but with a monthly tonnage of about 17,000 tons. Careful records of the performance of these hoists were kept on the forms shown in Figs. 5 and 6, and the results for a year's work plotted together with the shaft horsepower-hours (Fig. 7). A study of this curve will show that where the Ward-Leonard set has been kept working steadily, its efficiency exceeds that of the rheostatic set by about 5 per cent., but where the shaft horsepower-hours have dropped, the efficiency of the Ward-Leonard set has dropped with it, thus showing clearly the bad effect of intermittent

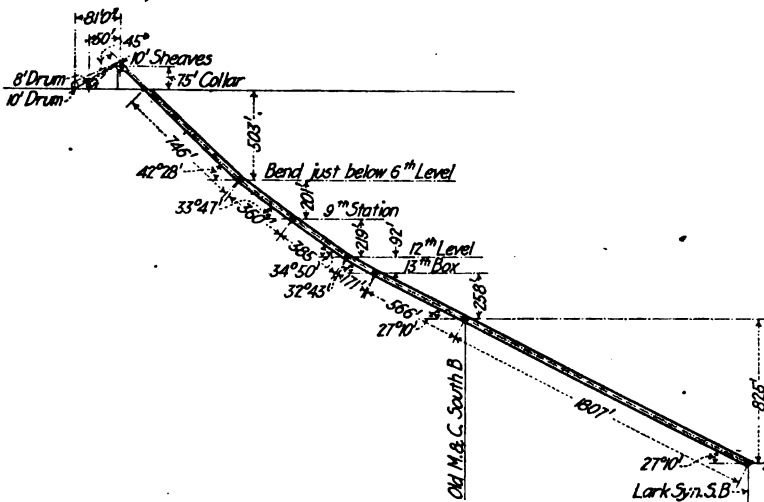


FIG. 4.—WARD-LEONARD WINDER, SECTION MAIN INCLINE SHAFT. MEYER & CHARLTON GOLD MINING COMPANY, LTD.

hoisting due to the practically constant losses of the motor-generator set, while with rheostatic control the only loss when the hoist is standing is that due to the controller pump motor which is negligible. The regularity with which hoisting can be carried on is well shown by the accompanying recording wattmeter and tachograph charts (Figs. 8 and 9) taken from the Meyer & Charlton plant. The wattmeter chart also shows clearly the effect of the dynamic braking in returning power to the line. Of course, to obtain such regularity it is necessary to provide suitable loading arrangements. For vertical shafts a modified form of the well-known Kimberley measuring loading chute has been found to answer well. For incline shafts, owing to the difficulty of quickly loading the skip full it was necessary to develop a form of measuring chute with a drop lip to deliver the ore well into the skip, and which in the closed position would

GENERAL MINING AND FINANCE CORPORATION, LIMITED.

ENGINEERING DEPARTMENT.

Miss Meger & Charlton

Month

Month April 1915

RECORD OF WORK DONE BY 5 Ton. Co. L. WINDER AT SHAFT: Mam. Machine

[illegible]

•Weight: 150 lbs. per Person

$$\text{Shaft H.P. hrs.} = \frac{\text{Total Tonnage} \times \text{Vertical depth in feet}}{\text{Shaft H.P. hrs.}}$$

Power Consumption 47880, K W hrs

FIG. 5.—PERFORMANCE RECORD OF WARD-LEONARD WINDER, MEYER & CHARLTON GOLD MINING CO., LTD.

GENERAL MINING AND FINANCE CORPORATION, LIMITED.

Wife

Kanby

ENGINEERING DEPARTMENT.

DEPARTMENT.
WINDER AT SHAFT. No 2

Month _____

Month - April 1915

RECORD OF WORK DONE BY 23/02

30

WINDER AT SHAFT

[illegible]

Weight: 150 lbs. per Person.

$$\text{Shaft H.P. hrs.} = \frac{\text{Total Tonnage} \times \text{Vertical depth in feet}}{550}$$

Power Consumption.. 40765... K.W. hrs

FIG. 6.—PERFORMANCE RECORD 5-TON HOIST, No. 2 SHAFT, VAN RYN GOLD MINING ESTATE, LTD.

keep this lip well clear of the skip way. Such a chute is shown in Fig. 10 and has answered very well.

COMPARATIVE COST OF ELECTRIC AND STEAM POWER

The writer would have liked to present tabulated statements of comparative costs of steam and electric driving of complete mining plants.

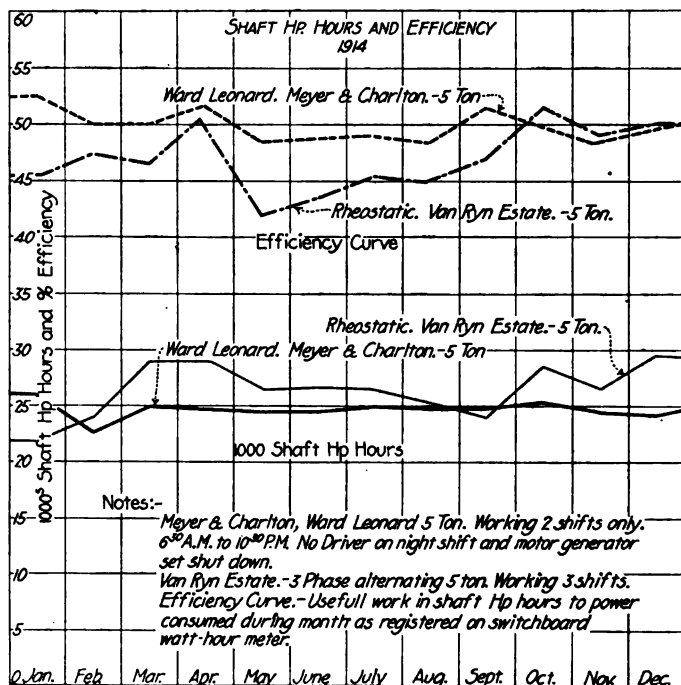


FIG. 7.—COMPARATIVE EFFICIENCY CURVE, WARD-LEONARD WINDER AT MEYER & CHARLTON, AND RHEOSTATIC AT VAN RYN ESTATE.

This was impossible as changes made in plants at the time of electrification would have made such comparisons seem unreliable except to those who had full knowledge of such changes. Also, in many cases steam had never been used. In one mine under the writer's charge, a complete change from steam drive to electric was made, the only boiler left in the plant being that supplying steam for cooking and wash water. In this particular case the question of the change was largely influenced by the fact that the mine was nearly at the end of its life. The acquisition of new ground gave the mine a much longer lease of life, and to continue with the existing steam plant would have meant a large amount of expenditure on power plant for renewals and repairs. For these reasons this installation could be regarded more or less as the equipment of a new mine. At the time this change was made the milling capacity was

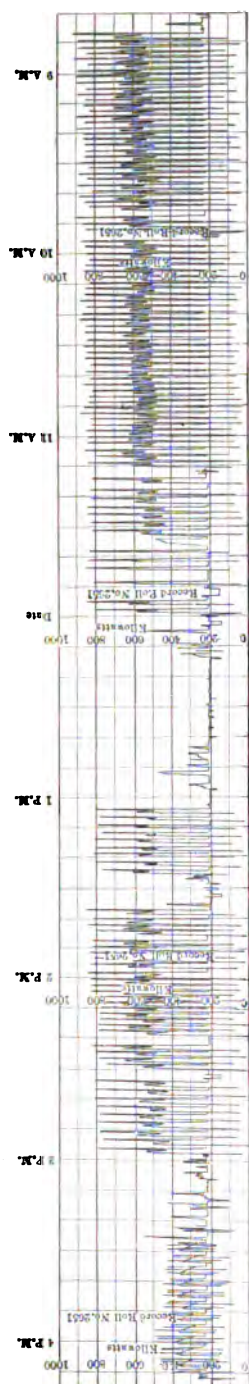


FIG. 8.—SECTION OF WATTMETER CHART, MEYER & CHARLTON PLANT.
Note.—Zero line has been raised to 200 to show braking power.

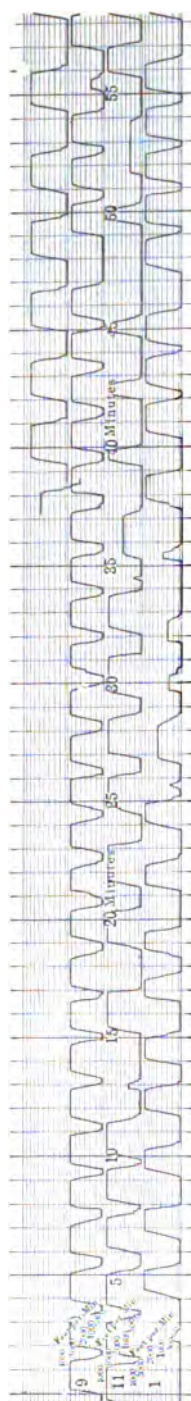


FIG. 9.—SECTION OF TACHOGRAPH CHART, MEYER & CHARLTON PLANT.

increased from 15,000 to 17,000 tons per month and a new compressor having 33 per cent. more capacity installed. A study of the steam power costs for the last two years the plant was in operation and electric power costs for a year's service showed the saving in favor of electric drive to be \$42,000 per annum without making any allowance for the increased power required by the additional tonnage treated and additional air supplied.

My thanks are due to E. Farrar, my former colleague in the General Mining and Finance Corporation, for his kindness in supplying the drawings and much of the information used in the paper.

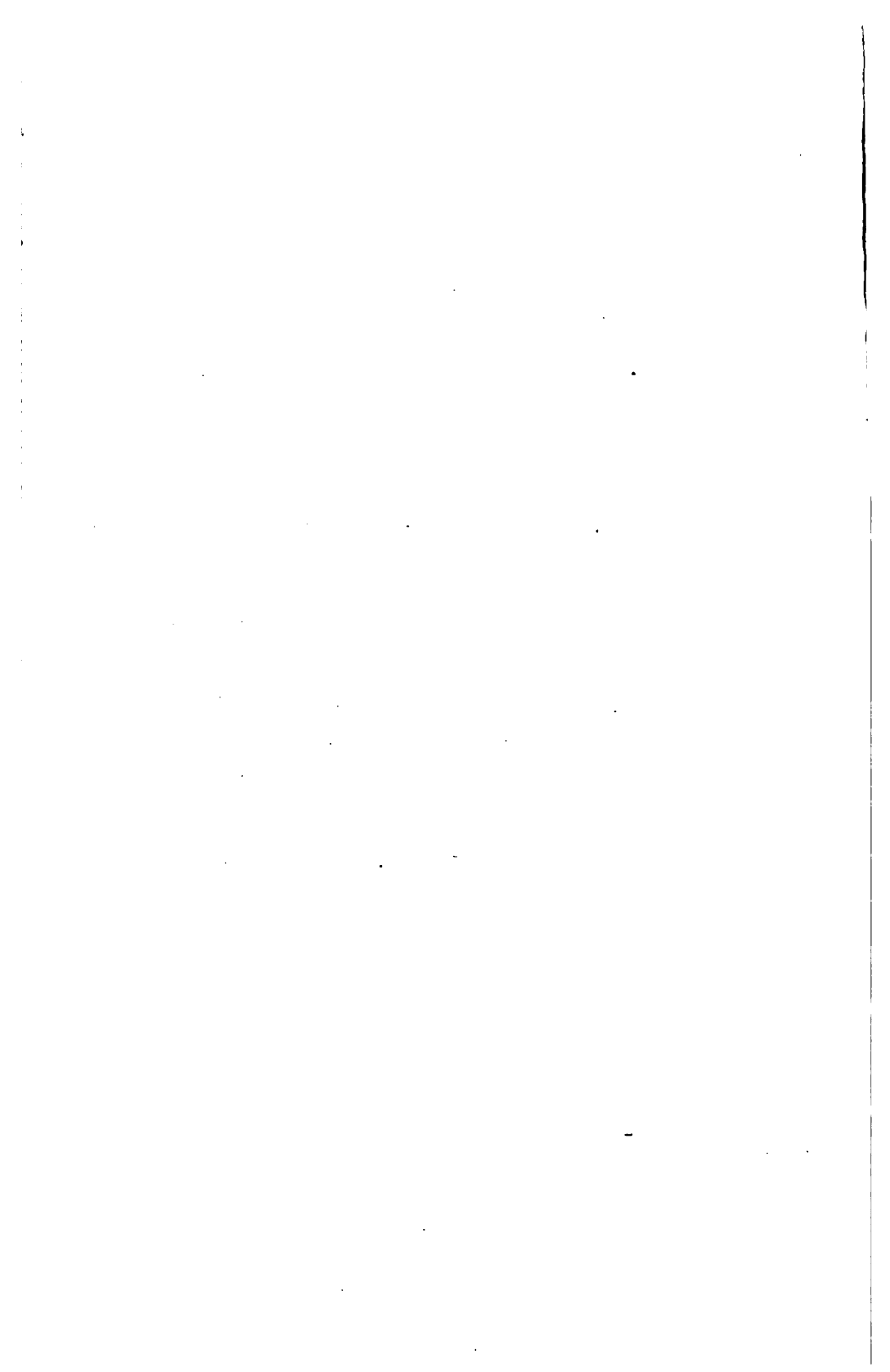
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 20 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Metallography of Steel for United States Naval Ordnance

THE REQUIREMENTS OF THE BUREAU OF ORDNANCE

BY HAROLD EARLE COOK,* PHILADELPHIA, PA.

(New York Meeting, February, 1916)

THE purpose of this paper is to state briefly the inspection requirements of the Bureau of Ordnance, the specifications governing the inspection, and the physical and chemical properties of the steel used in the construction of ordnance for our Navy.

It would be of interest to evoke discussion on points in these specifications which do not tend to insure the production of the very best material for the purpose intended. In other words, what requirements of our specifications are useless and also in what manner are our specifications deficient?

It might be well at the start to answer the first criticism of this paper which will probably be made, and that is that the title does not agree with the subject matter.

Considering metallographic work as covering only the macroscopic and microscopic examination of metals, this paper will have gone beyond the subject; but it is believed that a broader definition of metallography is that it is the study of the relation of the internal structure of metals and alloys to their composition and to their physical and other properties.

The internal structure of steel is dependent on its physical and its chemical properties. These physical properties are dependent on the chemical composition and the mechanical and thermal treatment of the steel. So it seems that this subtitle would not be entirely necessary and that the title "Metallography of Steel for Ordnance Purposes" would cover a study, not only of what we can learn of macroscopic and microscopic examination, but broadly of the whole subject of the specifications, the fulfillment of the requirements of which produce the structural conditions revealed by macroscopic and microscopic examination.

I will refer but briefly at the beginning to the microscope and its uses in our inspection. It is only very recently that the Bureau of Ordnance has furnished certain of its inspection offices with a portable Tassin outfit for the microscopic examination of steel. Since that

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time, the offices thus supplied have been studying the subject, and making considerable progress in order to keep up with the development of the art, so that when it reaches the state of development whereby it may be used to definite purpose in inspection, the various inspection offices will be equipped and prepared.

With the exception of our specifications for gun forgings there are no ordnance specifications at the present time which make reference to metallographic examination, as it may be said that defects which might be developed by routine microscopic examination would certainly be developed in the course of ordinary tests and inspection. Our specifications for gun forgings say: "The Department reserves the right to make such microscopic examination as it may deem necessary." The use of the microscope will be referred to in discussing the various specifications.

A few years ago each of the Bureaus under the Navy Department had its own specifications for the different classes of material required by it. For instance, as to steel forgings, the Bureau of Construction and Repair had its own specifications for material for forgings for hull material; the Bureau of Steam Engineering had its own specifications for engine forgings; and the Bureau of Ordnance had its own specifications covering forgings for mount work and mechanisms and gun forgings. Each Bureau had its own specifications for nonferrous alloys. This led to considerable trouble and complications with the various manufacturers; and recently the Navy Department has been endeavoring to combine, as far as possible, the specifications of the various Bureaus, getting them out as Navy Department specifications in leaflet form. A booklet of General Specifications is issued giving the requirements common to all material, method of testing and instructions to inspectors, which forms a part of each individual specification. These specifications are now used by the Bureau of Ordnance for ordinary forgings and castings, both of steel and nonferrous material for use in mount work, small parts of breech mechanisms and miscellaneous ordnance work. But for the more important ordnance material it is still necessary to issue separate and detailed specifications. Thus, there are separate detailed specifications for armor, projectiles, gun forgings, and torpedo air flasks and heads.

The specifications for armor are issued in booklet form prior to the award of each armor contract. The details of these specifications are confidential. The manufacturer is required to furnish the Bureau's inspector with reports of chemical analyses, etc., and the required amount of discard and forging reduction, as specified. The armor for each battleship is divided into groups of from 500 to 600 tons, which means 10 to 20 plates, depending on their size and thickness. As each armor plate in a group receives its final treatment, physical-test specimens are taken to show the properties of the plate, analyses are furnished, and other physical data; and when the last plate of a group receives its treat-

ment, one of the plates is selected for ballistic test. This plate is sent to the Naval Proving Ground at Indian Head, Md., and subjected to a ballistic test, being required to withstand the impact of a certain number of projectiles at given velocities.

The microscope plays no part in the inspection of armor, but this office has been making examination for purposes of study, of fragments of armor plate, and it is believed that some day information gained by the use of the microscope may prove of great value, at least in the manufacture of armor.

Specifications for the inspection of armor-piercing projectiles are also held confidential. As each lot of armor-piercing projectiles, when completed, is subjected to a ballistic test, a certain number of projectiles being selected from the lot for this purpose, great latitude is allowed the manufacturer as to the method of manufacture, discard, composition, etc.; but to promote uniformity in the lot, a maximum and minimum variation from the average of each of the important alloys used is prescribed. Physical tests are prescribed for the material used in the base plug of the shell, and also for the copper rotating band. Though it is believed that metallographic examination is of the greatest value to the manufacturer of armor-piercing projectiles, it is not thought that it will ever be of any practical value for inspection purposes, as the ballistic tests are such as to insure the excellence of a lot of projectiles which successfully endure the test.

In the manufacture of both armor and projectiles, every operation is open to the inspector, who is a commissioned officer.

Navy specifications for steel castings give requirements for six different grades. Of these, two grades in particular are used for ordnance material.

Grade "F," which is used for castings for gun yokes and special castings, must show a tensile strength of 85,000 lb. per square inch, a yield point of 53,000 lb. per square inch, with 22 per cent. elongation in 2 in., and 30 per cent. reduction of area. A bending test must give a 120° bend about a 1-in. diameter. It is required that this grade contain from 3.25 to 3.75 per cent. of nickel and that the phosphorus and sulphur content be not over 0.05 per cent.

Grade "B" is used for most ordinary castings and for those of great size. Its physical and chemical requirements are a maximum tensile strength of 80,000 and a minimum of 60,000 lb., with a yield point equal to 45 per cent. of the tensile strength obtained; elongation in 2 in., 22 per cent. and reduction of area 25 per cent. The same bending test is required as for Grade "F," and the phosphorus must not be over 0.06 nor the sulphur in excess of 0.05 per cent. The maximum and minimum tensile-test requirements are to insure proper machineability.

Castings for ordnance purposes vary from those of the very smallest size to massive castings, such, for example, as a slide for a 14-in. gun

mount, in which the complete casting, before machining, weighs in the neighborhood of 17 tons.

The specifications require that coupons from which test specimens are to be taken shall, whenever practicable, be cast on the body of the casting, and that, when this is not practicable, the coupon shall be cast with, or gated to, the casting, or with runners to the gate. The inspector's approval must be obtained in all cases, and the number of tests taken must be such as to exhibit thoroughly the uniform quality of the casting.

It has been my experience that tests taken from coupons gated to a casting are not indicative of the true condition of the body of the casting. The solidification of the metal may not be under the same conditions. The coupon would freeze more rapidly than the greater mass of metal. The segregation of the constituents would not be the same. The treatment effect might be quite different, especially if the requirements laid down for treatment in specifications, which will be discussed later, were permitted to be departed from. I believe that it would be much more desirable, where possible, to machine the test specimens from desired points in the body of the casting, but of course this procedure would be objected to strongly as increasing the expense.

With castings so small as to render it impracticable to cast coupons on the body of the casting, and where a number are made from the same heat and treated together, I believe that the preferable method of test is to require a sufficient extra number of castings to be made from each heat to select at random a number to be cut up for test. To cast, as is sometimes done, a coupon for test separate from the intricate-shaped castings, and to test it to pass upon a lot, or to test representatively castings differing in shape and size, gives to my mind, a false indication of the condition of the castings themselves.

In the larger castings it is the custom to cast coupons for test at various points on the cope and drag side of the casting, about 2 by 4 by 8-in. in size. On the castings as supplied to the Naval Gun Factory, coupons of this size form a portion of the pattern. These are cast so that the 4 by 8-in. area is fixed to the body of the casting, and they are used if at any time after delivery it be considered advisable to take additional tests from the castings.

As to the relative value of tests from coupons, I will cite an example. A manufacturer of some large, heavy castings, used coupons of approximately the size above named, but with the narrow edge of the coupon continuous to the body of the casting, so that the coupon could be easily knocked off, avoiding machining. Test specimens from these coupons gave satisfactory physical values and the castings were provisionally accepted. After delivery at the Gun Factory, additional tests were taken from some flat coupons which were machined off the castings; and so marked a difference was noted, both in all the physical properties and in

the microstructure as to preclude the use of these castings for the purpose intended. A number of them were re-treated and subsequently accepted on tests taken out of the body of the casting as furnished, no more coupon remaining. It was with great difficulty that the requirements were met.

An examination of the microstructure exhibited in the various test bars showed a wide variation in structure. The structures were nearly all pearlitic, but those from the original coupons showed a grain refinement not exhibited in the body of the casting at that time; and the pearlite was in a different state. These castings had been treated in a rather crowded casting pit, and it is thought that the coupons on edge projecting out from the body were more affected by the heat and that their rate of both heating and cooling differed from that of the rest of the casting. And further, it is possible that the loss of heat by radiation from the exposed surfaces of the coupon, exceeded the rate at which it could receive heat by conduction in cooling from the body of the casting. It is also probable that the furnace heat was insufficient to cause the body of the casting to reach the desired temperature for proper treatment.

In castings of which the shape and size would permit air- or oil-quenching, it is believed that the difference which I have noted would be very much more marked.

The specifications include a clause on treatment, in which it is specified that castings to be annealed shall be held at the temperature of annealing long enough to insure that the interiors of the castings have been brought to that temperature. After the castings have been soaked at the proper annealing temperature, they must be allowed to cool slowly in the furnace, carefully protected from drafts or air. Unless otherwise directed by the inspector, castings must not be removed from the furnace until they have been cooled down to the temperature at which the color dies. Under no circumstances will the manufacturer be allowed to use water, brine, oil or air-blast to hasten the cooling process, without the approval of the inspector. The number of hours requisite for raising the castings to the proper temperature, the length of time during which they should be soaked at that temperature, and the period required for slow cooling in the furnace, or in the air, may be prescribed by the Bureau concerned if it so desires.

Large castings are generally considered a cheap product in comparison with forgings, and appear never to have received the consideration as to scientific treatment and endeavor to give them a good structure that is bestowed upon high-grade forgings. It appears to be only quite recently that casting plants have begun to install pyrometric control for regulation of temperature. It appears to be still a disputed question whether a casting could ever be put in as good a condition as a forging. It is my belief that the most scientific treatment would fail to render a casting the equal of a forging as carefully made and treated. I believe that the

primary ingot crystallization can be completely destroyed only by means of mechanical working, or, if this theory is not correct, that the different structural conditions developed in the body of a massive casting in the ingot state, cannot be brought to a uniform condition by thermal treatment.

I am conducting, in a small way, some experiments with specimens of castings. I was first directed to this opinion by the fact that in numerous specimens examined microscopically, broad ferrite bands were noted from time to time extending across the field of the microscope, which did not appear to be in the nature of the usual "ghost lines" formed. It has occurred to me that the primary crystallization of the ingot which has crystals so large that they are often noted by the eye without magnification, or noted in macroscopic examination, are never completely destroyed. Heat treatment may dissolve the ferrite from the boundaries to such an extent that they are completely masked and cannot be seen by the microscope, or even if they are, the primary crystals may be so large that the usual field of magnification is contained in one large cell. This theory, of course, is not a new one, and has been the subject of discussion, the majority of authorities appearing to disbelieve it.

There are certain points open to discussion, which I will touch upon, but inasmuch as the treatment is largely left to the approval of the inspector, when a different logical treatment is desired for physical properties, his approval could be obtained. For instance, the temperature of 1,500° prescribed would manifestly give poor results if but a single treatment were attempted, as prescribed, with a grade "F" casting containing a rather high percentage of nickel and possibly of carbon, since long exposure at a temperature considerably above the upper critical would result in a large grain size. Many of the castings, specimens of which I have examined, show clearly the effect of this treatment.

In some cases large castings are placed in pits, heated up to a somewhat uncertain temperature, and held (at times) for as long as a day, and then cooled slowly in the furnace for an equal or longer time. The long exposure at high temperature is considered necessary to insure the soaking of the casting, but it certainly also insures a grain growth (provided the critical temperature has been exceeded). The extremely slow cooling through the critical range allows the rejection in totality of the ferrite, and in the low-carbon castings it often resolves itself into extremely large ferrite grains with large isolated pearlite patches in the grain boundaries.

It seems remarkable that physical tests from such castings give the results that they do, generally meeting the requirements. It is believed that impact or vibratory tests on this material would develop deficiencies in comparison with more scientifically treated material.

Some castings which have received a double annealing treatment



FIG. A.
Ingot steel, as cast. Carbon, 0.28 per cent. $\times 68$.



FIG. B.



FIG. C.—Ingot steel, as cast. $\times 21$.



FIG. D.—3.5 per cent. nickel steel,
as cast. $\times 68$.

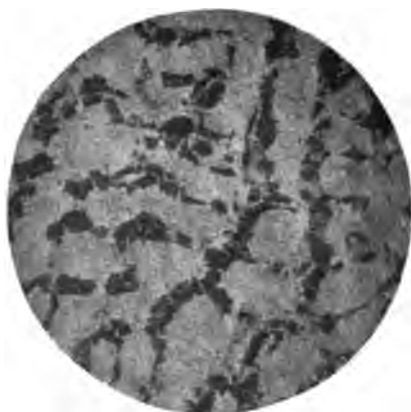


FIG. A.— $\times 80$.

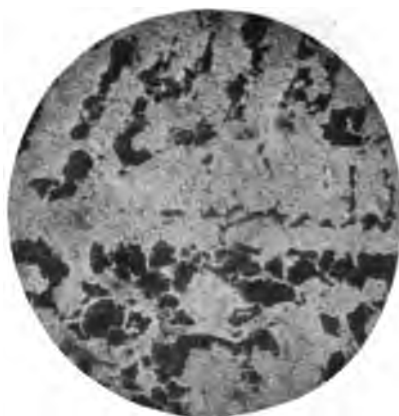


FIG. B.— $\times 80$.



FIG. D.— $\times 21$.



FIG. C.— $\times 21$.

PLATE II.—STEEL CASTING.

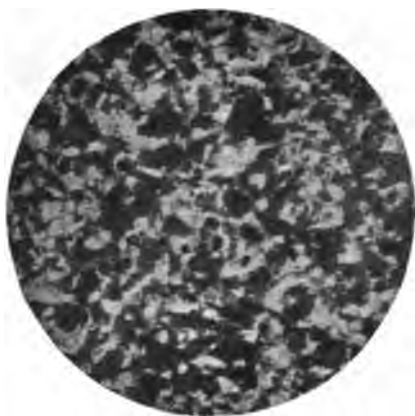


FIG. A.— $\times 80$.



FIG. B.— $\times 80$.

PLATE III.—NICKEL-STEEL CASTINGS.

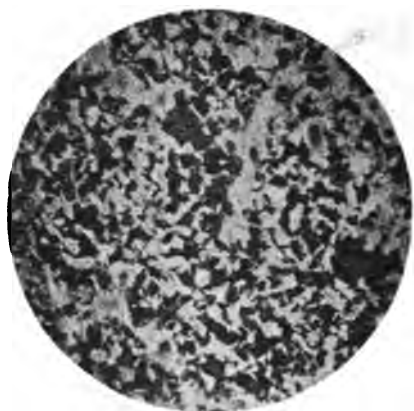


FIG. A.— $\times 80$.

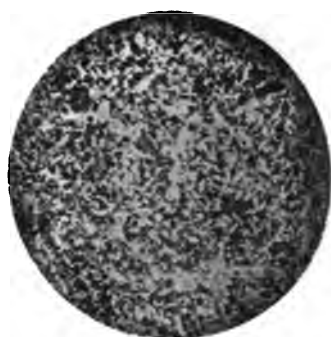


FIG. B.— $\times 32$.

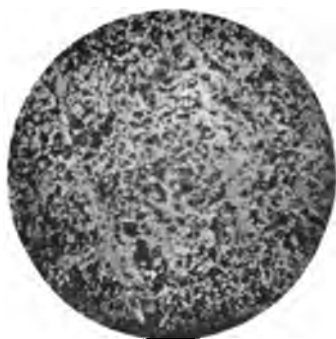


FIG. C.— $\times 32$.



FIG. D.— $\times 80$.

PLATE IV.—STEEL CASTINGS.



FIG. A.— $\times 80$.



FIG. B.— $\times 80$.



FIG. C.— $\times 80$.



FIG. A.—0.30 per cent. carbon. $\times 80$.

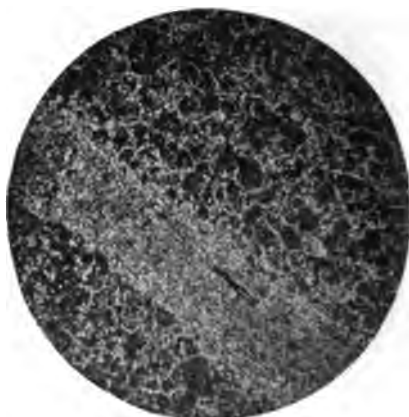


FIG. B.—0.40 per cent. carbon. $\times 80$.

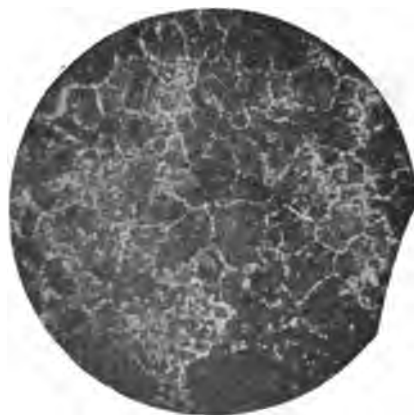


FIG. C.—0.42 per cent. carbon.

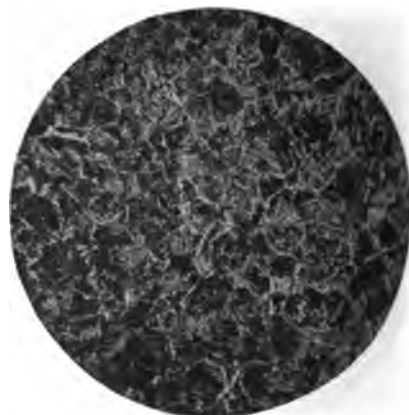


FIG. D.—Carbon, 0.40 per cent; nickel, 2 per cent. Annealed, quenched, drawn. Tensile strength, 96,500 lb. per square inch; elastic limit 66,000 lb. per square inch; elongation in 2 in., 22.5; 59.9 per cent. reduction in area.



FIG. A.
Microstructure of test specimens at breech end. $\times 80$.

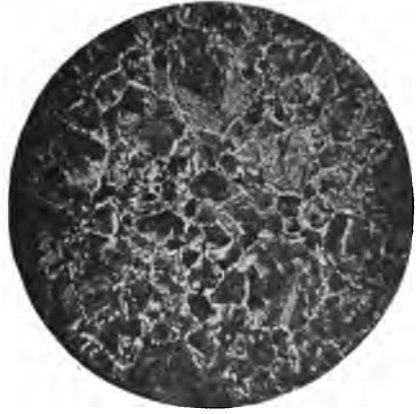


FIG. B.

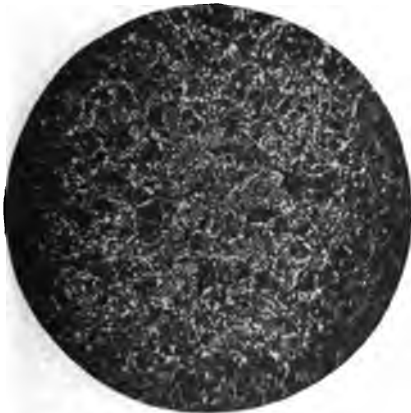


FIG. C.

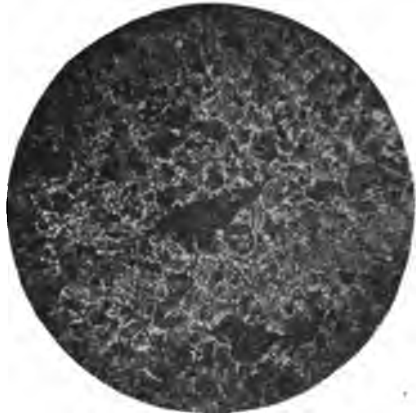


FIG. D.

Microstructure of test specimens at muzzle end.

PLATE VII.—GUN FORGING, ANNEALED, QUENCHED AND DRAWN. CARBON, 0.30 PER CENT.; NICKEL, 3 PER CENT.

ALL MET REQUIREMENTS: TENSILE STRENGTH, 90,000; ELASTIC LIMIT, 60,000; ELONGATION IN 2 IN., 18; REDUCTION IN AREA, 30. FIG. C GAVE BEST RESULTS, AND FIGS. A AND C GAVE HIGHEST PERCENTAGE REDUCTION OF AREA.

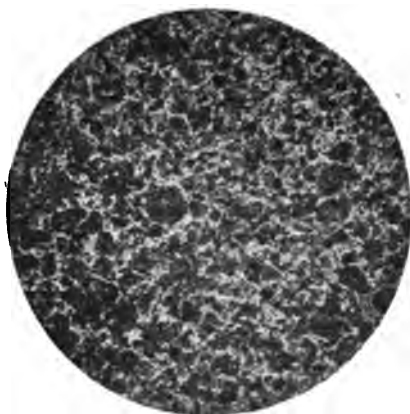


FIG. A.—Nickel steel. Carbon, 0.40 per cent. Tensile strength, 95,000; elastic limit, 62,000; elongation in 2 in., 24; reduction of area, 50. $\times 80$.

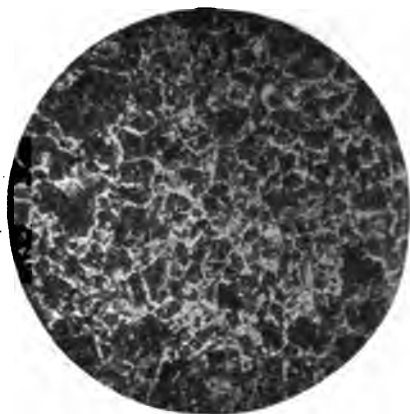


FIG. B.—Nickel steel. Carbon, 0.40 per cent. Tensile strength, 112,000; elastic limit, 70,000; elongation in 2 in., 21; reduction in area, 48. $\times 80$.



FIG. C.—Small nickel-chrome gun forging. Carbon, 0.40 per cent. Tensile strength, 104,000; elastic limit, 68,000; elongation in 2 in., 22.9; reduction in area, 58. $\times 82$.



FIG. D.—Rare smaller forging. Tensile strength, 105,000; elastic limit, 63,000; elongation in 2 in., 25; reduction in area, 54. $\times 155$.

PLATE VIII.—CHARACTERISTIC MICROSTRUCTURES FOUND IN NICKEL STEEL FOR GUNS—ANNEALED, QUENCHED AND DRAWN.

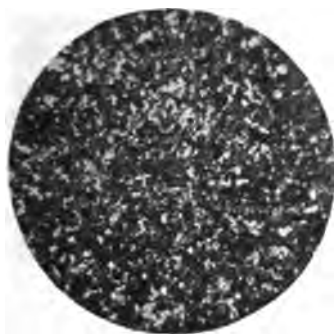


FIG. A.—Small forging of 3.5 per cent. nickel steel. 0.35 per cent. carbon. Tensile strength, 95,250; elastic limit, 60,000; elongation in 2 in., 24; reduction in area, 54. $\times 82$.

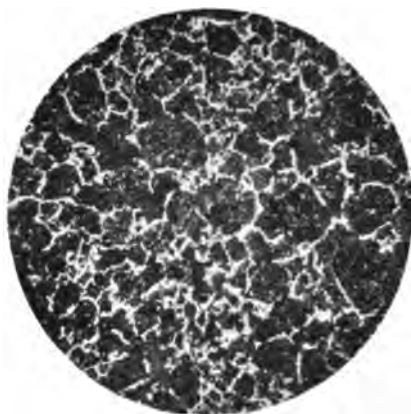


FIG. B.—Gun steel. 0.50 per cent. carbon. Tensile strength, 98,000; elastic limit, 58,000; elongation in 2 in., 18; reduction of area, 30. $\times 80$.

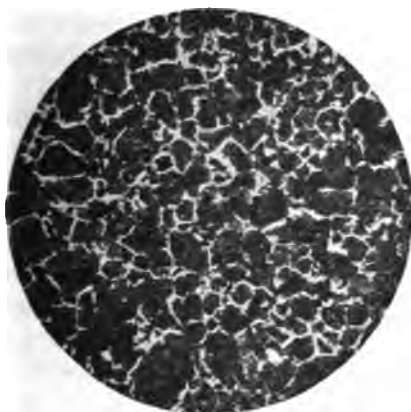


FIG. C.—Gun steel. 0.45 per cent. carbon. Tensile strength, 99,000; elastic limit, 57,000; elongation in 2 in. 25; reduction in area, 54. $\times 80$.

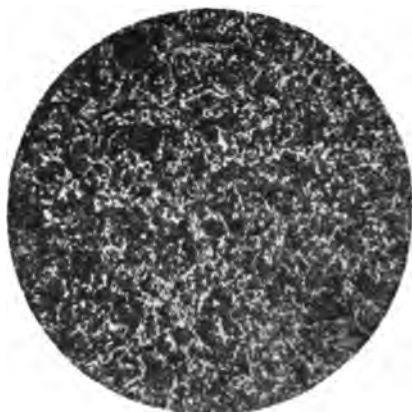


FIG. D.—Gun steel. 0.50 per cent. carbon. Tensile strength, 105,000; elastic limit, 63,000; elongation in 2 in., 26; reduction in area, 53. $\times 80$.

PLATE IX.—CHARACTERISTIC MICROSTRUCTURES FOUND IN STEEL FOR GUNS—
ANNEALED, QUENCHED AND DRAWN.

have exhibited excellent results. It is believed that if greater care were exhibited in the heat treatment of castings to insure heating them and holding them at just above the upper critical, taking into careful consideration the chemical composition, and cooling them at such a rate through the critical range as will give the most desirable properties, physical properties vastly superior to those at present obtained would be the result.

I will now give some examples of the microstructure found in various ordnance castings. The specimens examined are prepared by sawing off the test specimens transversely in the threads and polishing and etching by the usual methods.

Plate I shows the structure of a class "B" casting in the green, as cast, and before any heat treatment. Fig. C shows a complete ingot crystal at a low magnification (21). Fig. D shows the structure of a casting containing 3.5 per cent. nickel, in the green.

Plate II shows some photographs taken from physical test specimens from a Class "B" ordnance casting weighing about 500 lb. This casting was accepted on the test specimen, which showed a tensile strength of 73,500 lb., a yield point of 44,500, with 24 per cent. elongation in 2 in., and 38.7 per cent. reduction of area. It was a simple steel casting of about 0.24 carbon which had received an annealing treatment from a temperature supposed to be just above its critical range.

Figures A and B show the structure at 80 magnification. The totality of the ferrite appears to have been rejected, and the pearlite under high magnification shows the normal laminated structure.

Figures C and D show the same structure at a magnification of about 21, including a greater area in the field. It will be seen that the structure resembles very closely an ingot structure which has apparently not been broken up by the treatment. It is thought that the treatment temperature was probably less than reported, and was not high enough or sustained long enough to break up completely the original structure. It seems remarkable that such satisfactory physical results should have been obtained.

Plate III shows two specimens taken from 0.30 carbon, nickel-steel castings which contained between 3.25 and 3.5 per cent. of nickel and about 0.2 per cent. of chromium. These castings received a double annealing treatment and showed from 93,000 to 102,000 lb. tensile strength with a yield point of from 61,000 to 64,000 lb., with 22 per cent. elongation and 38 to 40 per cent. reduction of area. In the structure of these castings it appears that the alloys have caused a part of the ferrite to be retained in solution, as the dark areas are not pure pearlite, appearing to be in the transitory stage with some sorbite.

Plate IV shows a structure taken from another physical test specimen of practically the same composition, which gave a tensile strength of

91,000 lb., a yield point of 64,000 lb., elongation of 23 per cent., and reduction of area of 41 per cent. In this case there appears to be a greater amount of ferrite and its segregation appears in the nature of a ghost. The pearlitic structure is not characteristic, as the pearlite grains are so small.

Figure *A* shows the structure at a magnification of 80, and Figs. *B* and *C* at 32. The ghost lines are most prominent in the lower magnification.

Figure *D* shows the same specimen after it had been heat treated in the small furnace in my office. It was heated in one hour to a temperature of 1,500°F., held at this temperature for one-half hour and cooled in the furnace in two hours. The ghost-line structure seems to have disappeared, and the ferrite is more evenly distributed. It is not known, of course, what physical properties would be exhibited by such a structure.

In Plate V, Fig. *A* shows the structure exhibited by a test specimen of a class "B" casting, which passed physical test, giving the following results: tensile strength, 70,500 lb.; yield point, 36,000 lb.; elongation, 27.5 per cent.; and contraction, 49.2 per cent.

This was a 0.26 carbon, simple steel casting, and had been held at a temperature supposed to be 1,650°F. for 22 hours and cooled for 17 hours. It presents the characteristic structure of an ingot, which is noticeable by comparison with Plate I. It is not understood how such good results were obtained in elongation and reduction of area with such a poor microstructure.

Fig. *B* shows a similar casting which gave similar results. The upper critical temperature was probably never reached in the treatment of these castings.

Fig. *C* shows the structure of a 0.30 carbon, nickel-steel casting containing about 2 per cent. of nickel. The physical results shown were: tensile strength, 89,000 lb.; yield point, 54,500 lb.; elongation, 18 per cent.; and reduction of area, 29.2 per cent.

Long heating in a single treatment, and slow cooling, evidently caused the ferrite to collect in large grains.

From the very poor structure exhibited in the examination of numerous ordnance castings, I believe that great progress will be made in future in the development of more satisfactory and uniform castings through more scientific heat treatment. Probably at the present time the specification requirements are none too severe, so that with well-melted steel the fulfillment of the requirements is not a difficult matter.

Castings received and finally accepted for installation in the manufacture of ordnance, appear to be highly suitable for the purpose intended. It does not appear to be advisable to incorporate requirements which would compel the manufacturer to install expensive equipments

for more scientific treatment, thereby increasing the cost. But a special grade might be added to the specifications which would cover these points.

At present the greatest difficulty in the development of ordnance castings appears to be the preventing of blow holes, segregations and shrinkage defects.

As regards the microstructure noted in the examination of ordnance castings, it may be said in general that a pearlitic structure is exhibited; that when an alloy steel is used the totality of the free ferrite is not rejected and there is a predominance of the darker areas. The usual impurities are noted throughout the structure always in the form of rounded areas, the slate-colored impurities being silicate of manganese, whereas the manganese sulphide has the characteristic dove color.

Before taking up the more important question of forgings for ordnance purposes, it may be of interest to remark that while the Bureaus of Steam Engineering, and Construction and Repair have chemical laboratories, where analyses are made of drillings taken from castings and forgings by their inspectors, the results to be passed on before acceptance, the Bureau of Ordinance has not. A copy of all analyses taken by the manufacturers is always required, and this has been considered sufficient. Check samples have been taken from time to time, and in cases of doubt, and sent to the nearest government chemist for analyses. The results have not seemed to warrant making our own analyses of all material. This course is probably rendered doubly safe by the fact that nearly all ordnance material of importance is subjected to a ballistic test of some nature before final acceptance, and much latitude is therefore allowed the manufacturer in the chemical composition used.

The specifications for steel and alloy-steel gun forgings, except for a few very minor details, are the same for both the Army and the Navy. They are issued in the form of a pamphlet, revised from time to time, which goes into the requirements for manufacture and inspection in considerable detail.

The manufacturer is required to furnish reports of all analyses and physical tests made by him during the manufacture. He is required to furnish reports in advance of each operation in the manufacture in order that any operation desired may be witnessed. The government stamp is placed on each forging, transferred as necessary, and carried through to completion.

The term "gun steel" is defined as a simple carbon steel, and the term "alloy steel" as a simple carbon steel with such percentage of alloy added as is necessary to obtain the required physical qualities. Thus, if the required physical qualities could be obtained with a simple carbon steel, the manufacturer would be at liberty to do so, and the question has been considered from time to time of specifying a minimum allowable percentage of nickel in alloy steel, in order to impart certain desired

properties and to prevent a too rigid heat treatment to obtain the required properties. At present the only definite requirements as to composition are that the allowed percentage of sulphur and phosphorus shall not exceed 0.05. In examination of the microstructure of gun forgings, sulphur is usually seen in the form of manganese sulphide, which will be discussed later, and phosphorus segregations have been suspected in certain light-colored areas which have been brought out by the use of Rosenhain's reagent.

The specifications require that from open-mold ingots the discard from the top shall be at least 30 per cent. of the total weight of the ingot, and from the bottom at least 5 per cent. Possibly it would be better if the specifications did not prescribe a definite amount of discard but required the discard in each case to be sufficient to obviate any chance of piping or marked segregations. It is my opinion that 5 per cent. discard from the bottom of large ingots is an insufficient amount, but as this would be developed in the testing, the amount in excess of 5 per cent. should be left to the discretion of the manufacturer. From fluid-compressed ingots there is a slight reduction in the amount of discard required.

It being understood that the object of forging, in addition to its necessity for shaping a piece, is also to impart to it a mechanical grain refinement to put it in a condition whereby heat treatment will attain the desired end, a minimum amount of forging reduction is prescribed. The specifications say that ingots must be of such size that when pieces are forged solid the area of cross-section of the ingot shall be at least four times the maximum area of cross-section of the rough forging, and when bored ingots are used, the wall thickness of the ingot must be reduced by forging 50 per cent. Some forgings are of such a size that it would not be possible to make them from the largest ingot and give them the amount of reduction prescribed by the specifications. Other forgings are of different size at each end, and the shaping of them of necessity requires a different amount of forging reduction. In many such cases it is necessary in the process of manufacture that the ingot, after being punched or bored, be upset a certain amount, and in other cases forgings are drawn out and then enlarged on a bar. In all such cases a record is made of the total amount the metal has been worked. In other words, the upsetting operation is also considered as a forging reduction, though it is not, of course, in strict accordance with the wording of the specifications. It must be recognized that the amount of forging reduction alone will not insure a mechanical grain refinement, and that the temperature at which the forging is completed, and the subsequent treatment of the forging, play a most important part. If the forging is stopped at too high a temperature and the metal allowed to cool slowly, there will, of course, be a grain growth, and in large masses of metal this cannot be entirely prevented, as the metal cannot be worked entirely down to the

critical range; but this is taken care of in the requirements under "Heat Treatment." Physical tests and metallographic inspection of the ends of forgings which have had a different forging-reduction at each end, have not shown any marked difference attributable to the amount of reduction. Most long forgings will show the best physical results in specimens parallel to their axes, and short hoops enlarged on a bar transversely. But most forgings which are worked in more than one direction mechanically show little difference between transverse and radial or longitudinal bars.

The specifications for treatment require that the manufacturer keep a record and furnish copies of same to the inspector, showing the length of time for heating, the time held at that temperature and the time of cooling of every operation in thermal treatment. It is also required that he have an accurate means of ascertaining the temperature used. It is also required that, when practicable, forgings must be rough machined before treatment. This is necessary to insure maximum penetration and uniformity in the treatment. The precautions to prevent too rapid heating and the length of time during which the forging is held at annealing temperatures, is required to be satisfactory to the inspector.

There is a provision also in the specifications requiring all forgings to be annealed from the temperature above the Ac_1 point after forging and before tempering. This provision for the preliminary annealing covers the point referred to above, but it is believed that the Ac_3 critical point should be prescribed in place of the Ac_1 to insure better results.

It is required that all gun forgings of length be heated for quenching in vertical furnaces, which it is believed insure a more even and uniform heat, and the forgings are required to be immersed in the quenching medium in the direction of their axes. Longer pieces such as tubes and some hoops are often distorted a slight amount in treatment and it becomes necessary, before final machining, to straighten them. The specifications allow no straightening after test for acceptance as to physical properties. It is also required that the straightening shall be performed at a temperature between 800 and 900°F. and that after straightening they shall be reheated to the same temperature and annealed.

I have seen physical tests made of some shafting, and also some gun forgings, before and after straightening at this temperature, and though these tests showed no difference in physical properties, yet reheating is thought to be a wise precaution, since probably at this temperature the metal is more pliable. I believe tests have been conducted on metal slightly heated which have shown a falling off in physical properties at that temperature, which are apparently restored when the metal is again cooled.

When ample allowance of test metal on each end of a gun forging is allowed by the manufacturer, and when, after heat treatment, trial-test bars show the forging to be in proper condition, it is submitted to the

inspector for official test. The test specimen used is a standard cylindrical bar 0.505 in. in diameter and 2 in. between measuring points. The number of test specimens taken from a forging varies. From the largest forgings, four specimens are taken from each end and a smaller number for the smaller sizes. The number of test specimens taken from an end should be a function of the area of cross-section and not of the length. An exception to this would be in the case of a very short ring in which it might be considered not necessary to test each end. Thus, if three tests from each end are required from a hoop of a certain length, there would be no occasion for taking more than three tests from each end of another forging, the length of which is two or three times as great; that is, assuming that three tests from an end are sufficient to demonstrate its uniform quality. All bars from gun forgings subject to bursting stresses are taken circumferentially, or rather tangentially.

It is required that the minimum distance of the axes of test specimens from the end of a forging shall be 1.5 in. for long forgings and 1.25 in. for all other forgings. It has been found necessary to prescribe a distance, because tests taken close to the face give a slightly different property where the metal has been quenched, the rate of cooling probably being affected by the radiating surface of the end.

The physical properties prescribed (tensile strength and elastic limit in pounds per square inch; elongation on 2 in. and reduction of area in per cent.) vary with the type of forging. For alloy-steel forgings they vary from 90,000, 55,000, 18 and 30, for tubes, to 95,000, 55,000, 18 and 30 for certain hoops. For gun steel they vary from 86,000, 46,000, 18 and 30 to 93,000, 53,000, 18 and 30.

It is believed that it would be very desirable to increase the requirements for reduction of area. In my opinion, that reduction of area is one of the most desirable requirements inasmuch as it is an indication of the amount of distortion which will result before rupture after the tensile strength has been reached, and it also shows the material capable of sustaining a fiber stress greatly in excess of the tensile strength before actual rupture of the metal occurs.

As will be shown later, the mass of large gun forgings is such that it is practically impossible to eliminate segregations and defects such as ghost lines or streaks from the forging. If the steel were pure, there would not be much difficulty with the modern scientific treatment in meeting the requirements of the specifications as to physical properties. But while nearly all the test specimens from a forging may give results considerably in excess of 40 per cent. reduction, there may be one or more bars which will give less, caused by the aforesaid conditions, so that the increase in this requirement would probably be objected to strongly by the manufacturer, unless some provision were made to take care of bars which might fall short.

In figuring the elastic limit and tensile strength in tests, reference is always made to the original cross-sectional area which in the test bar used is 0.2 sq. in. Thus the stress per square inch is obtained by multiplying the actual load on the beam of the testing machine by 5. Even at the point when the tensile strength has been reached, there has been a slight reduction of area so that the actual sustained load on the test bar at that point has been greater than the recorded tensile strength. In the same manner if the actual sustained load on the beam could be measured after the test bar has necked down and just before rupture, it would be found to be very much greater, due to this reduced area. This sustained load at rupture, or rupture stress as it has been called, seems to me to be of extreme value, and it would be of value to develop some practical method of determining and recording it.

The elastic limit is required to be determined by the use of the micrometer extensometer. This requirement may be waived, provided the testing machine used is shown to have no difference between the elastic as determined by the drop of the beam and the micrometer readings for the grade of material in question. Of course this would only be true in machines where friction and inertia of moving parts are inconsequential.

Loads below the elastic are required to be applied in accordance with a table in the specifications, and for each load the actual extension must be recorded. It is required that the accuracy of the testing machine shall be verified at least once each year.

Three official submissions for physical tests are allowed on each forging, which may be either without or following retreatment. In case one bar from an end on physical test fails in extension and breaks within 0.5 in. of the measuring point, or in case one bar from an end breaks and the fracture shows that it failed because of inclusions of sand or slag, an extra bar is allowed immediately in wake of, and nearer the finished metal than the failing bar.

The tensile specimens are the only ones taken for physical tests from material for gun forgings, and are all that have been considered necessary, in view of the fact that the final test consists in the proof firing of the gun. In the test of material used in foreign gun construction, it is understood that the physical properties are generally lower than those prescribed for ours, but certain other tests are required in addition, such as mandrel tests, bending tests and fatigue tests. These last tests, when autographic vibratory machines reach perfection, may prove of great value and may demonstrate the unsuitability of material which would not be shown by other tests.

During the past year or so, for the purpose of collecting information, we have been examining and photographing the structures of many gun forgings. The test specimens are usually sawed off in the threads and the button thus formed is polished and etched in the usual manner. A

number of characteristic structures found in what are considered to be excellent forgings, are shown in Plates VIII and IX.

The use of nickel appears to insure a finer grain as well as to prevent the precipitation of the same amount of ferrite, that would result in plain gun steel.

Microscopic and macroscopic inspection has proved of the greatest value in investigating the character of defects in large forgings which are often noted in machining. These defects have been referred to in the specifications as follows:

"In the case of streaked forgings, that is, forgings showing striations of different-colored metal, careful examination will be made to see if there is any lack of continuity of the metal along such streaks. Such lack of continuity determined by the breaking or parting of the chip in turning or in chiseling in the direction of the streak, or in any other manner, will cause the rejection of the forging.

"Slag pockets and sand splits or cavities containing particles of slag, sand, or other foreign material will be treated as local physical defects in the steel, the seriousness of which will depend upon their location in the forging, their number per unit area of surface, their size and form, etc. If isolated and small their presence in a forging otherwise sound and satisfactory will not be deemed important; if in clusters or extending over a considerable surface, indicating a porous condition of the metal, the forging will be rejected."

Defects commonly known as streaks and sand splits exist to a certain extent in all large forgings, where the ingot from which they have been made has been one of considerable mass, but every effort is made toward reducing them to a minimum.

The defects known as sand splits are detected only in the inspection of surfaces smooth machined, and are found generally in the bore of large forgings after the last machining operation. They vary in size from the most minute specks to small pockets as long as $\frac{1}{2}$ in. At times larger ones have been observed. These splits, which are generally elongated in the direction the metal has been most worked in forging, contain inclusions which often in the larger ones may be picked out with a small penknife, and resemble sand. Some of them contain a harder inclusion more difficult to remove. They are certainly related to the microscopic inclusions of sulphides and silicates which are generally segregated in ghost lines or streaks.

In Plate VI, Fig. A, is shown, at a magnification of 80, a microscopic inclusion of such a size that it appeared on the polished surface of the specimen to the eye without magnification, as a pin point. This inclusion appears to contain three substances. One, which is slate-colored (the darker in the photograph), is believed to be silicate of manganese. The matrix, which is dove-colored, appears to be manganese sulphide. There is a third constituent (white in the photograph), which is not understood. This specimen was from a piece of extremely impure steel; and the dark boundary of the inclusion which appears in the photograph

is a cavity caused by ripping out a portion of the inclusion in polishing. There are also shown numerous spots of sulphide.

Streaks appear to be of two general classes. They are noted as faint lines in the bore and on the surface of machined forgings, where the metal appears slightly brighter or darker in color. Polishing the surface and etching with iodine solution in alcohol will bring them out clearly as will also sulphur prints.

Sulphur prints, which are often made to investigate forgings where segregations of sulphides are suspected, are made by polishing carefully the surface to be examined, cleaning it off very carefully with alcohol to remove all oil, then placing upon it, face down, a piece of bromide paper which has been immersed in a solution of hydrochloric acid. After a few seconds the acid will attack any segregations of sulphides, discoloring the paper, and the photograph so formed is then fixed in the usual hypo solution. These sulphur prints bring out clearly the streaks when the inclusions which predominate in the streak are sulphides. Many streaks and inclusions are noted, however, in forgings when no clear sulphur prints of them can be obtained. These inclusions, which are generally slate-colored, are hence believed to be silicates. It should be noted that a sulphur print can be obtained from the polished surface of any piece of steel inasmuch as the sulphur in the steel, if almost entirely in the form of manganese sulphide evenly distributed, will cause an impression on the sulphur print of many specks which may appear to be more serious than the condition warrants. When these specks are examined after etching with the usual solutions, under the microscope, their character is clearly brought out.

In the first class of streaks which I have noted, there is a marked predominance of ferrite and an extremely fine grain. On each side of the streak the grains increase in size until the normal sorbitic structure is noted. Scattered along the streak are generally found many microscopic inclusions of sulphides and silicates often in lines. The metal in these streaks does not appear to differ from that on either side of it as to machining, and scleroscope readings do not show any difference in the hardness. When the inclusions are arranged in lines they, and any lines or cracks joining them, are carefully examined.

In the worst of these streaks, where the inclusions are in such number and of such size as to cause an actual break in the continuity of the metal, their seriousness would be indicated by chipping out the metal and the minute breaks would be shown by examining with a small magnifying glass. The general character of one of these streaks is clearly shown in Plate VI, Fig. B, in which a line of very small inclusions is also shown. This was a small streak which extended through a physical test specimen which had passed test; and the only noticeable effect was a falling off in

reduction of area as compared to other bars from the same forging which did not show the streaked metal.

In the second class of streaks which I have noted, the grains are much larger and there is not so much ferrite predominating as on either side of the streak. On some of these streaks a difference in the scleroscope reading is noted and the metal is shown to be harder. The inclusions noted are of the same nature in both classes of streaks. These streaks are more broken and irregular. Their characteristic structure is shown in Plate VI, Fig. C. The small test specimen which this structure represents, exceeded all the physical requirements.

The question of streaks and sand splits is being constantly studied and all such defects, even though very slight, are made a matter of record. The final test of the gun of which these forgings are made, of course, is the proof-firing.

The photographs of streaked metal are of streaks of the less serious nature. In the more serious streaks, which are rare, the metal will break off in machining and actual breaks in the continuity of the metal will be noted on macroscopic inspection.

In Plates VIII and IX are shown characteristic average structures found in gun forgings. These structures in general consist of sorbite and ferrite intimately mixed and are logical structures following the treatments which they have received. Some structures appear to be pure sorbite, and almost amorphous, but these are generally the smaller alloy-steel forgings such as breech blocks and short hoops for the smaller guns. Gun forgings to meet the properties required after the preliminary anneal specified are quenched and drawn.

In Plate VI, Fig. D, also in Figs. A to D, Plate VII, is exhibited a peculiar structure which is often noted in gun forgings. The structure consisting of a network of ferrite and sorbite, shows much of the ferrite in definite parallel lines within the cells, in some cases having an almost triangular formation. At a high magnification this structure resembles much the structure of an ingot at a low magnification. It is thought to be due to the structure formed in quenching, running from troostite to martensite according to the composition, which has persisted after drawing, allowing the ferrite on rejection to mass along the lines of this structure. The physical test specimen from which Fig. D, Plate VI, was taken, gave excellent results.

Before closing this paper I will refer briefly to one other class of forgings which has caused concern in drawing up requirements, namely, the air flask of the torpedo. This forging is about $8\frac{1}{2}$ ft. long and 21 in. in diameter, and contains the air chamber for propelling the torpedo. It has to sustain a very high internal pressure and of necessity must be as light as possible. For the test of the forging, two tensile tests are taken from each end. We have obtained a number of forgings which have

passed specifications requiring 120,000 lb. elastic limit with 12 per cent. elongation in 2 in., and 35 per cent. reduction of area. Various specifications on other contracts have varied from this to 105,000 elastic limit with 15 per cent. elongation and 40 per cent. reduction. To meet such specifications an extremely pure steel is necessary. It has been noticeable in such tests as I have witnessed, that a small and inappreciable amount of slag may cause a test bar to break off short in this material, whereas the same amount of impurities in material in a softer condition would have an inappreciable effect on the results obtained by testing. This point is thought to be an important one, which should be considered by designers.

In conclusion, I might say as regards metallographic inspection, that at present in the larger inspection offices we have a Tassin microscopic outfit which can be applied to forgings in the shop if desired, or set up in our laboratory for examining smaller specimens there. This microscope has a photographic attachment and we have a dark room so that we may develop our own photographs as taken. It is our general practice in examining specimens for polishing to examine before etching, then to etch lightly and examine, and then to etch deeper and examine. In examining under the microscope we generally get as big a field, or low magnification, as possible at first and then examine with our standard magnification of about 100; and if interesting points are noted we then examine with higher magnifications.

We have a portable polishing apparatus which we can carry into the shops and polish surfaces desired.

We also have a scleroscope which we use generally for experimental purposes.

It will be seen that our aim at present in using the microscope in inspection is to collect all the data possible and to gain additional information regarding defects which may be noted in the usual inspection. It appears that the microscope is continually finding new uses to the manufacturer and more manufacturers are taking it up each day. It is believed that it has a big future, both to the manufacturer and later, probably, in inspection.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Observations on Certain Types of Chalcocite and Their Characteristic Etch Patterns

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(New York Meeting, February, 1916)

IN February 1913, Prof. L. C. Graton and Dr. Joseph Murdoch¹ presented to the American Institute of Mining Engineers a notable contribution to economic geology under the title *The Sulphide Ores of Copper*. The value of the work was recognized by mine operators, and Mr. Graton was assisted in his plan to extend his investigations, by the organization of a commission to undertake a "secondary enrichment investigation." In this work he secured the cooperation of the Geophysical Laboratory of the Carnegie Institution of Washington. Their report² on the chemical phases of the investigation has just been published.

Incidental to investigations³ carried on in my laboratory by advanced

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¹ *Trans.*, vol. xlv, 26 to 81 (1913).

² Posnjak, Allen, and Merwin: *The Sulphides of Copper*, *Economic Geology*, vol. x, pp. 491 to 535 (1915).

³ J. J. Beeson: *The Disseminated Copper Ores of Bingham, Utah*, *Bulletin No. 107*, pp. 2191 to 2236 (November, 1915).

J. C. Ray: *Paragenesis of the Ore Minerals in the Butte District, Montana*. I. The Covellite Zone, *Economic Geology*, vol. ix, No. 5, pp. 463 to 481 (July, 1914). II. Final paper in press (*Economic Geology*).

A. F. Rogers: *Upward Secondary Sulphide Enrichment and Chalcocite Formation at Butte, Montana*, *Economic Geology*, vol. viii, pp. 781 to 794 (December, 1913), (with W. H. Turner), *A Geologic and Microscopic Copper Sulphide in Plumas County, California, and its Modification by Ascending Secondary Enrichment*, *Economic Geology*, vol. ix, No. 4, pp. 359 to 381 (June, 1914). *Secondary Sulphide Enrichment of Copper Ores with Special Reference to Microscopic Study*, *Mining and Scientific Press*, vol. cix, pp. 680 to 686 (October, 1914). *Sericite, a Low Temperature, Hydrothermal Mineral*, *Economic Geology* (in press). *The Chemical Composition of Bornite*. *Science*, New Series, vol. xlii, pp. 386 to 388 (1915).

C. F. Tolman, Jr.: *Recent Advances in the Study of Sulphide Enrichment*, *Mining and Scientific Press*, vol. cviii, pp. 172 to 176 (January, 1914) (with J. D. Clark), *The Oxidation, Solution, and Precipitation of Copper in Electrolytic Solutions and the Dispersion and Precipitation of Copper Sulphides from Colloidal Suspensions, with a Geological Discussion*, *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (September, 1914). *Microscopic Study of the Ores of the Bonanza Mine, Alaska* (in manuscript).

students and myself, and along mineralogical lines by my colleague, Prof. A. F. Rogers, we have collected data that seem to have a bearing on points raised in the papers mentioned. It was planned to delay the publication of certain of these results until further information and more definite conclusions had been obtained. However, the chemical data presented by Posnjak, Allen, and Merwin, bridge in part the gaps in our work, and the interest aroused by their research justifies the publication of a preliminary statement of the results of investigations not as yet completed.

In addition to accurate descriptions of the occurrences and structures of the copper sulphide minerals, Graton and Murdoch ventured to suggest a definite criterion to distinguish chalcocite formed from ascending solutions from that deposited by descending solutions. They believed the former to possess "a good cleavage,"⁴ further defined as "three cleavages at right angles," and the latter . . . "usually shows, instead of cleavage, countless irregular cracks . . ." These closely spaced cracks commonly form the outlines of individual grains . . .⁵ These structures, of course, are developed by etching. If this criterion could be established and readily applied it would have great practical value.

These suggestions of Graton and Murdoch are modified by the investigators of the Geophysical Laboratory as the result of their exact research work. They verify the suggestion of Hittorff⁶ that cuprous sulphide is dimorphous. They find it to be orthorhombic below 91°C. and isometric above this temperature. Artificial chalcocite, formed at high temperatures shows, on etching, an isometric cleavage, equally well developed along three planes mutually perpendicular to each other. Orthorhombic chalcocite, on the other hand, shows a basal "etch cleavage." According to my observations, the etching of that orthorhombic chalcocite which is an aggregate of minute crystals develops the outlines of the irregular but roughly equi-dimensional crystals, each crystal having a fine lining or striation in one direction (Figs. 26 and 34). Coarsely crystalline orthorhombic chalcocite, however, is characterized by an etch pattern which is stronger and more regular in one direction than in the other directions (Figs. 38, 39 and 40).

The chemical investigations mentioned above show that both artificial and natural cuprous sulphide may contain cupric sulphide in solution, a fact also discovered by Hittorff in 1851. If more than 8 per cent. of cupric sulphide is present no inversion point from one form into the other was detected. If, therefore, chalcocite containing more than 8 per cent. of CuS is formed above 91°C. it should retain its isometric cleavage. Chalcocite from the Bonanza mine, Alaska, contains more than

⁴ *Loc. cit.*, p. 54.

⁵ *Loc. cit.*, p. 57.

⁶ *Poggendorff's Annalen der physik und chemie*, vol. lxxxiv (1851).

the requisite amount of CuS , and it is suggested, on account of its perfect isometric cleavage (Figs. 6 and 7), that it is "primary," and formed above 91° . My microscopic investigation does not verify this conclusion.

A discovery resulting from the chemical investigations carried out in the Geophysical Laboratory is that the blue color of chalcocite increases with the amount of cupric sulphide in solution. Of interest in this regard, I find chalcocite of pronounced blue color of the following three types: (1) In small veinlets cutting bornite, chalcopyrite and pyrite, especially near the top of the zone of sulphide enrichment; (2) as borders around residual masses of bornite. This blue border is evidently an intermediate product between bornite and white chalcocite and contains fragments down to sub-microscopic size of residual bornite; (3) as complicated patterns (Figs. 9, 10 and 11) in what we have designated in the laboratory as "two-colored chalcocite." The best examples of the latter were found in the ores from the Bonanza mine, Alaska. On account of its supposed isometric (high temperature) etch cleavage, its high content of cupric sulphide in solid solution, and the mooted question as to the genesis of the enormous masses of pure chalcocite outcropping at the surface, the Bonanza chalcocite is of especial interest. In this paper I will show that it has been derived from bornite (Figs. 4 and 12); that the cleavage is inherited and is oriented with the two-colored patterns of the chalcocite; that the patterns are pseudomorphic after certain stages in the replacement of bornite, and therefore, in general, by these patterns we may find clues as to earlier minerals which subsequently may have been completely destroyed by the processes of chalcocitization.

I also present below examples of the control which the original bornite structure retains over the development of chalcopyrite from bornite (Figs. 19 and 20), of blue and white chalcocite from bornite, and even over the development of malachite from chalcocite which in turn is secondary after bornite (Fig. 26).

Graton and Murdoch⁸ recognized that bornite, breaking down along crystallographic lines, transmitted its etch pattern to chalcocite, but believed that the latter could be distinguished from true isometric cleavage of chalcocite. My studies indicate that the pseudomorphic etch pattern may develop great regularity (Fig. 27), and that a triangular or

⁷ The authors appear to give the word "primary" a geological meaning; that is, they use it to designate ores formed from ascending solutions. In this paper, I use the terms "primary" and "secondary" in a strictly mineralogical sense, and therefore a statement, for example, that one sulphide is secondary after an earlier one, carries with it no hypothetical inference as to whether the ore-forming solutions were ascending or descending.

⁸ *Loc. cit.*, xlv, p. 58.

rectangular pattern (Figs. 5, 6, 7 and 8) in natural chalcocite, in which all three directions of cleavage are equally regular, proves it to be a replacement of an earlier mineral, generally of bornite. I have found the two types of etching in a single grain of chalcocite (Fig. 28), a beautiful isometric etch figure in the center where the original bornite structure controls, and a fine mosaic of striated orthorhombic crystals on the outside, where the replacement destroys instead of preserves the original bornite structure.

A type of chalcocite (Figs. 29 to 36) is figured and described below which has not been recognized by Graton and Murdoch or by the investigators of the Geophysical Laboratory. Microscopic study indicates this to be a meta-colloid. It occurs in the upper portion of the sulphide zone, and all occurrences studied so far suggest that it is formed by descending solution. It shows a characteristic and extremely minute etch structure (Fig. 34). The texture and porous character of meta-colloidal chalcocite, the manner in which it replaces pyrite, and the structure it develops upon etching are shown in photographs in (Figs. 29 to 37).

To illustrate the foregoing points I have selected as far as possible ores from deposits, the geology and paragenesis of which have been carefully studied. For examples of blue and white chalcocite (Figs. 1, 9, 10 and 11) I have had to use ores from the Kennecott-Bonanza mines, Alaska, although heretofore the ore has been wrongly described as primary, and the earlier cycles of mineralization have not been recognized. This makes it necessary to summarize briefly the results of my microscopic study of this ore. The control of bornite structure through successive generations of minerals (Figs. 18 to 28) is well shown in ores from the Apache mines, Pima county, Ariz. Meta-colloidal chalcocite is figured from Bingham, Utah (Figs. 29, 30 and 31), and the disseminated deposits of Miami, Ariz. (Figs. 36 and 37). Two examples (Figs. 32, 33 and 34) from museum specimens, (locality unknown) are included, as they are the most beautiful and instructive specimens of the meta-colloidal type of ores that I have seen.

In order to make the data as complete as possible, the geological and mineralogical history of the ores is described, and in certain cases rather definite evidence is presented as to the temperature at which the minerals were formed and as to the source of the mineralizing solution, whether from below or from above.

Blue and White Chalcocite from the Bonanza Mines, Alaska

The ore deposits at the Kennecott-Bonanza mines in southeastern Alaska are unique in that they contain enormous masses of practically pure chalcocite outcropping at the surface. If we assume that these masses of chalcocite represent the extreme concentration of sulphide

ores developed by the processes of downward enrichment, we are confronted by the fact that the oxidized zone, in which the enriching solutions had their origin, has been completely removed. Under present climatic conditions, oxidation, solution and reprecipitation are so slight that in most of the ore their results can only be recognized by the microscope, and they produce impoverishment rather than enrichment. However, the theories as to the origin of this orebody have been based largely on the belief that the chalcocite is "primary." Lindgren⁹ states, ". . . there is no evidence that it" (chalcocite) "has replaced pyrite, and it is in all probability of primary origin." According to Moffit and Capps,¹⁰ ". . . no evidence was found . . . to indicate that the orebody has ever been anything other than what it is at present. The copper sulphides appear to have been deposited as such, and a careful examination of the ore has failed to discover the presence of other minerals than those produced by the alteration of the chalcocite."

The Group of Early Minerals.—Microscopic study of ore specimens^{10a} Theodore Chapin, of the United States Geological Survey and Stephen Birch, President of the Kennecott Copper Corporation, shows that there is an early generation of minerals, the remains of which are now shown by corroded residual microscopic specks. These are bornite (Fig. 4), galena, and klaprothite (?) (Fig. 1). The last two occur in minor amounts and by far the greater portion of the chalcocite is derived from bornite.

Blue and White Chalcocite of the First Generation.—That the Bonanza chalcocite is derived from bornite is proved by its intricate patterns in blue and white (Figs. 9, 10, and 11). The writer has observed that in ores from many localities, triangular patterns of blue and white chalcocite form only where bornite has suffered disintegration along its planes of cleavage in the initial stage of chalcocitization. Furthermore, examination of the Bonanza ores under high magnification was successful in discovering residual stringers of bornite (Figs. 8 and 12). These are arranged in rows, and these rows represent the last remnants of large crystals of bornite destroyed by chalcocitization. The rows, in general, are parallel to the patterns in blue and white, characteristic of the first generation chalcocite, and to the isometric etch pattern inherited from the bornite.

It appears certain, therefore, that the Bonanza chalcocite is the result of an unusually complete alteration of a great coarsely crystalline mass of bornite, and that a million tons or so of iron have been completely re-

⁹ *Mineral Deposits*, p. 383 (1913).

¹⁰ *Geology and Mineral Resources of the Nizina District, Alaska. Bulletin 448, U. S. Geological Survey*, p. 82 (1911).

^{10a} The photographs of the Bonanza ores were taken during the preliminary study of a small suite of specimens presented by Theodore Chapin and Stephen Birch. Extensive collections have since been received from the U. S. Geological Survey and from Dr. J. F. Newsom, and up to date, the conclusions presented above need no serious revision.

moved from the orebody and an equivalent amount of copper has been added by the processes that have caused this transformation.

A second pattern commonly developed (Fig. 11) seems to be pseudomorphic after the so-called "ice cake" structure. The latter is produced by the attack on the surfaces of each bornite crystal with little or no eating in along cleavage lines. This leaves irregular residual masses of bornite set in a matrix of chalcocite (Fig. 13).

These patterns are rarely noted on freshly polished surfaces but in short time (30 sec. in one case) the design begins to develop, and within a day or two reaches a maximum contrast, after which there often appears to be a slight fading of the color contrast. It would appear that the color is due to the unmixing of the solid solution which develops submicroscopic crystals of covellite.¹¹

White Chalcocite of the Second Generation.—This appears as veinlets in the older blue and white chalcocite, and around crystal boundaries. This later generation of chalcocite is shown best by etching with potassium cyanide. In Fig. 5 are seen the large crystals of blue and white chalcocite with etch pattern inherited from the bornite, between which veinlets of white chalcocite of the second generation are injected. In Fig. 6 the parallel striations of the second generation of chalcocite contrasts strongly with the regular isometric etch pattern of the first generation of chalcocite. These later veinlets connect directly with the series of veinlets filled with malachite, and along which rosettes of melaconite are developed (Fig. 14), proving that this later chalcocitization is the result of the oxidizing solutions passing through the orebody today. Along the chalcocite veinlets, covellite needles (Figs. 15, 16, 9 and 10) "eat out" into the older chalcocite crystals. A further stage in the alteration results in the complete penetration of the large crystals of chalcocite by covellite (Fig. 15) and the transformation of the blue and white chalcocite with regular etch pattern into white chalcocite with an irregular fine etch pattern (Fig. 7). This is an example of the unmixing of the solid solution, blue chalcocite into white chalcocite and covellite, brought about by late supergene action. Chalcopyrite is developed in connection with covellite. At high magnifications specks of chalcopyrite are seen at the points of the covellite needles (Fig. 16). The chemistry of the process is plain. A small amount of iron remains as an impurity in the early chalcocite. It is a legacy from the bornite stage. The covellite needles grow into the chalcocite pushing the iron along in

¹¹ The investigation of the Geophysical Laboratory proves that the blue color of the chalcocite increases with the amount of CuS in solution. It does not follow necessarily, that this is the only cause for the various colors of chalcocite. For another explanation of the blue chalcocite of the type that occurs in the complicated patterns, see article by Young and Moore cited above.

front of the crystal points until it is present in sufficient amount to force the formation of the chalcopyrite.

Summary

The paragenesis of the ore minerals of the Bonanza deposit may be summarized as follows:

First Group of Minerals.—Bornite, klaprothite (?) and galena. Temperature of formation probably relatively high for copper deposits, as Prof. Rogers¹² has found klaprothite, bornite and chalcocite to be grouped together in most of the chalcocite deposits thought to have been formed by ascending solutions, and at temperatures above those at which downward enrichment takes place.

Second Group of Minerals.—Blue and white chalcocite of the first generation, secondary after bornite. The temperatures governing, and the source of the solution causing this alteration are unknown.

Third Group of Minerals.—This includes a set of minerals forming progressively under conditions of increasing oxidation and represented as follows:

Second generation of white chalcocite $\rightarrow \left\{ \begin{array}{l} \text{covellite and} \\ \text{chalcopyrite} \end{array} \right\} \rightarrow \text{tenorite}$
 (and cuprite) $\rightarrow$ malachite $\rightarrow$ azurite. This group is formed at about 0°C., as much of the ore is frozen throughout the year. It is the result of the present vadose circulation, and the copper is migrating largely as malachite (Fig. 3).

As to the structures of the Bonanza chalcocite, the two-colored patterns are portraits of earlier stages in the breakdown of bornite to chalcocite. The regular cleavage, instead of proving a primary origin of the chalcocite, indicates a secondary origin.

The original ores were coarsely crystalline bornite with accessory minerals such as galena and klaprothite (?). The process by which these have been changed almost completely into chalcocite belongs to a cycle of ore deposition which is not active at present.

The present low-temperature vadose circulation is developing the third group of minerals mentioned above, and results in an impoverishment rather than an enrichment of the ore.

The above conclusions should be taken into account in the formulation of any theory as to the origin of the first generation of chalcocite. I leave to Prof. Graton (who has had every opportunity to study the ore and the mine) the task of deciding as to whether the extreme chalcocitization that has taken place at the Bonanza mine was the result of descending solutions acting in the past under warm climatic conditions, or whether the

¹² Unpublished manuscript.

chalcocite, undoubtedly secondary in the mineralogical sense, was the product of ascending solutions, as affirmed by Sales for the Butte ores.

Ores from the Apache Mines, Santa Catalina Mountains, Pima County, Arizona

Ores from this mine were chosen to illustrate the breakdown of bornite along crystallographic lines, and the preservation of these directions in minerals of subsequent generations. The geology and mineral paragenesis of these deposits has been worked out in detail by the writer during the mapping of the Tucson quadrangle for folio publication for the United States Geological Survey. The salient features are summarized below in order to show clearly that the first formed copper minerals, chalcopyrite and bornite, were deposited at elevated temperatures, and that the complex breakdowns and replacements described in detail are the result of alterations which are taking place at or near the surface.

Summary of the Genesis of the Ores

The deposits are of contact metamorphic origin, formed at and near the contact of a quartz diorite sill with Paleozoic limestone. The series of events which culminated in the formation of workable ores are:

1. The intrusion of a quartz diorite sill and smaller sills of albite diorite porphyry.

2. Development of contact metamorphic silicates and ores, especially in those localities where the sills contain lenses and segregations of acid pegmatitic material. This is of importance in showing the relation of the ores to the processes of magmatic differentiation. These minerals are grouped according to their order of formation as follows:

Group I. Garnet, green pyroxene and magnetite. These are very high temperature minerals.

Group II. Pyrite, epidote, quartz, molybdenite, chalcopyrite bornite, all cut by tremolite. Temperature of formation moderately high, as tremolite is the latest mineral of the group, and here is an after effect of contact action. The copper minerals replace pyrite in part. Minerals of this group cut and replace minerals of the first group.

3. Oxidation and enrichment of the orebody.

Group I. Chalcopyrite (second generation), blue and white chalcocite, covellite. These are formed locally near the top of the orebody by descending solutions; ore unimportant. Temperature 0°–20°C.

Group II. Chrysocolla and indefinite mixtures of copper oxides and carbonates forming at or near the surface. Temperatures 0°–25°C.

Pegmatite dikes and high-temperature quartz-epidote veins (or dikes) cut the orebodies, and are believed to have been injected between the formation of mineral groups I and II. Basic dikes cut the orebody and are undoubtedly later than all the contact minerals. The first two

groups of minerals, therefore, fall within the period of the activity of magmatic process connected with the intrusion of the quartz diorite.

The Structures of Minerals Secondary After Bornite.—Special attention is directed to the decomposition of the bornite. The alteration is undoubtedly here due to superficial solutions.

Fig. 17 shows garnet cut and replaced by chalcopyrite and bornite, and Fig. 18 the high-temperature chalcopyrite and bornite cut by, and earlier than, the tremolite needles. The bornite in this slide has been attacked by solutions locally rich in iron and the second generation of chalcopyrite develops along the cleavage lines of the former. This type of replacement of bornite is only found in an orebody formed by replacement of a basic sill, and the source of iron is therefore at hand. Fig. 19 is a higher magnification of a much smaller speck of bornite and shows the details of this alteration. Fig. 20 shows chalcocite replacing bornite more readily than chalcopyrite leaving the latter as a residual pattern in the chalcocite. Fig. 22 presents a typical rectangular structure developed by a partial replacement of bornite by chalcocite along a veinlet, and Fig. 23 a similar regular pattern preserved in blue and white chalcocite with residual specks of bornite. Fig. 24 shows the second generation of chalcopyrite which has replaced bornite and, in turn, is attacked by chalcocite which redevelops structural lines parallel to the cleavage of bornite. Such structures were not developed by chalcocitization of the first generation of chalcopyrite. Fig. 25 shows covellite replacing bornite along cleavage lines. This is from the Engels mine, Shasta County, California described by Rogers.¹³ It is given as another example of the control of the bornite structure on later alterations. Finally, Fig. 26 represents malachite replacing chalcocite along directions inherited from bornite. This example is important in showing that preservation of structures does not involve a crystallographic orientation of the minute chalcocite crystals with the bornite structure and cleavage, and therefore, that the pseudocleavage is not true cleavage of chalcocite. The individual crystals are shown in the blue and white patterns of the chalcocite and have a fine orthorhombic striation similar to that developed by etching and described below.

As shown in Fig. 27, a regular etch pattern was developed by treating blue and white chalcocite with nitric acid. This chalcocite is similar to that shown in Fig. 23, and, of course, is pseudomorph after bornite. Fig. 28 shows both the regular etch pattern after bornite and an irregular pattern characteristic of chalcocite. The latter was developed just outside the area in which the bornite structure controls.

The control shown by the bornite structure through one or even two successive generations of minerals is merely an example of successive pseudomorph replacement which the reflecting microscope has shown to be common in ore deposits. The best examples known to the writer of

¹³ *Loc. cit.*

structures inherited through several generations of minerals are shown in the complicated ores of Butte, Mont. Beautiful examples are figured in J. C. Ray's final article¹⁴ on the Butte ores.

It is a strange fact that alteration takes place along crystallographic planes in some cases and in other cases it attacks the crystal surfaces without regard to crystal planes and produces rounded irregular structures such as the "ice-cake" structure (Fig. 13) and the so-called "graphic intergrowths."

Summary

The ore deposits at Apache camp, Ariz., are contact deposits, the unenriched ores consisting of chalcopyrite and bornite and very minor amounts of pyrite. These are known to have been formed at high temperatures, intermediate in time between the first and last minerals produced by contact action.

The bornite especially is unstable under conditions of local oxidation and enrichment near the surface, and forms chalcopyrite (where solutions are rich in iron) blue and white chalcocite, covellite, and malachite. In the center of each speck of bornite the breakdown is always along the cleavage planes of bornite, but on the margins this structure may not control. The secondary minerals inherit the bornite cleavage as a replacement structure and not a true cleavage. Etching of secondary chalcocite re-develops the perfect isometric cleavage of bornite, especially in the center of the grains while the margins may show the more irregular cleavage or parting characteristic of chalcocite. It is suggested that a very regular isometric cleavage in natural chalcocite may be a proof of the secondary nature of the same, the antecedent mineral being bornite.

Meta-colloidal Chalcocite

As is well known, artificial precipitates of metallic sulphides are colloidal. Experiments by J. C. Clark¹⁵ have shown that in the presence of hydrogen sulphide both natural minerals and sulphide precipitates are dispersed in so finely divided a condition that they pass through pores and fine openings like true solutions. They pass through filter paper and are precipitated by the escape of hydrogen sulphide. Later experiments by Young and Moore¹⁶ show that the sulphides of copper can be taken into solution by hydrogen sulphide and re-precipitated in crystalline form with the utmost ease.

In case the precipitate is first a gel, this may crystallize slowly, and the resultant mosaic of fine crystals may preserve the structure of the

¹⁴ *Economic Geology*, in press.

¹⁵ Tolman and Clark: The Oxidation, Solution and Precipitation of Copper in Electrolytic Solutions and the Dispersion and Precipitation of Copper Sulphides from Colloidal Suspensions, with a Geological Discussion. *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (September, 1914).

¹⁶ *Economic Geology*, in press.

original gel. For such crystalline substances Wherry¹⁷ introduced the term "meta-colloid."¹⁸

If, as suggested by Tolman and Clark, fine dispersion by hydrogen sulphide is the important factor in the transportation of metallic sulphides, and if precipitation of sulphides by the escape of hydrogen sulphide is equally important in ore genesis, the recognition of colloidal structures in sulphide minerals is of interest. Such structures might be expected near the surface and in open spaces rather than in the deep-seated ores which have undergone slow and thorough crystallization and repeated re-crystallizations by replacement. As far as known to the writer, meta-colloidal chalcocite has not heretofore been described.

Chalcocite ores from small veinlets from Bingham, collected by Prof. Rogers, showed upon examination a marked mammillary structure, typical of meta-colloidal minerals. Examination of a polished section of this ore revealed that malachite and the copper oxides were developing along conchoidal structural lines that otherwise would not have been recognizable. Figs. 29 and 30 are from this specimen and Fig. 31 shows a similar fine structure developed by etching. Later it was found that imperfect examples of this structure are common in ores developed near the top of the sulphide zone. Familiarity with this curved line and branching structure assists one to recognize this type of chalcocite by the curved and branching natural fractures, and the application of the etching test described below will settle doubtful cases. Fig. 37 shows a veinlet of this type of chalcocite replacing pyrite.

This type of chalcocite is unusually porous. Chalcocite ores in general are porous and thus ready access is given to circulating solutions of all kinds, but meta-colloidal chalcocite shows extreme development of porosity, so that it is polished with difficulty, and it is necessary to develop a thick flow surface to get a brilliant reflection. Porosity is a characteristic of many meta-colloids. Figs. 32 and 33 show a typical meta-colloidal chalcocite at different magnifications. Fig. 35 is a concretionary chalcocite inclosing quartz grains. The dust-like specks are the pores, the size of which can be compared with the very fine sand grains.

Chalcocite directly replacing pyrite, near the top of sulphide-enriched orebodies, especially in ores of the "disseminated type," show extensive development of the "exploding bomb" structure due to replacement in the interior of the crystals of pyrite along crystallographic faces. This is accompanied by a development of excessive porosity in the chalcocite deposited around the crystals of pyrite, and, to a less extent, between the crystal faces, so that the outlines of the original pyrite crystals are

¹⁷ Variations in the Composition of Minerals: *Journal of the Washington Academy of Science*, vol. iv, pp. 111 to 114 (1915).

¹⁸ For an excellent summary of data regarding mineral colloids and extensive bibliography see: R. Marc and A. Himmelbauer: *Fortschritte der Mineralogie, Kristallographie und Petrographie*, vol. iii, pp. 11 to 62 (1913).

plainly marked by cloudy chalcocite (see Fig. 36). It would appear that the first precipitate of chalcocite around the pyrite crystals and along the crystal planes opened by solution is colloidal, as is the material artificially produced with pyrite as a precipitant, and the setting (in this case, crystallization) of the gel further fractures the crystal and assists in its complete destruction.

All the above-mentioned examples of chalcocite suggestive of an original colloidal structure are characterized by an orthorhombic etch figure, consisting of aggregates of exceedingly minute crystals, each showing a typical striation in one direction. This structure is so fine that a high-power oil immersion lens is necessary to resolve it. Fig. 34 shows the etch pattern of the typical meta-colloid material figured in photographs 31, 32 and 33.

General Conclusions in Regard to Etch Figures

The following conclusions may be considered tentative to be expanded or modified as further data are collected.

No isometric etch figures have been found in natural chalcocite, that are not inherited after some antecedent mineral, generally bornite. Posnjak, Allen, and Merwin suggest that the Bonanza ore is the only natural example known of original isometric etch cleavage. I have shown this to be secondary after bornite.

Regular isometric inherited structures are found in chalcocite formed presumably by ascending solutions and certainly by descending solutions, and therefore cannot be used as criteria to determine whether the chalcocite is hypogene or supergene.

Orthorhombic etch structure with one direction cleavage, or parting distinctly more regular than other directions, is found wherever the structure of the mineral replaced does not govern. It is found in chalcocite formed presumably by ascending solutions (example: the so-called "graphic intergrowth" from Virgilina district, Virginia (Fig. 38) and ore showing "ice cake" structure from Butte (Figs. 39 and 40) and from supergene ore. It is formed where chalcocite replaces bornite or other minerals, without regard to the internal structure of the mineral replaced.

The development of a very fine orthorhombic etch structure consisting of minute individuals each lined with parallel striations (Fig. 34) is suggestive of meta-colloidal chalcocite. This is probably formed at the top of orebodies and is presumably deposited by descending solutions.¹⁹ Good examples are found in the "concretionary" chalcocite of Butte, the rich chalcocite veinlets at Bingham, and the "disseminated ores" of Ray and Miami, etc.

All gradations exist between the very fine meta-colloidal etch structure, and coarser, more irregular structure, characterized in general, how-

¹⁹ This is the fish-scale structure of A. Perry Thompson: *The Covellite Ores of Butte*, *Bulletin* No. 100, p. 656 (April, 1915).

ever, by one direction of fine striation for each crystal. Both of these differ from the more regular patterns, such as shown in Figs. 39 and 40. In the majority of cases the irregular type (including the fine meta-colloidal type) appears to be the result of descending solutions acting near the top of the orebody. In many cases a second generation of chalcocite has the irregular pattern and a first generation has the regular pattern.

Graton and Murdoch²⁰ undoubtedly had in mind these two types of pattern when they described the "primary" chalcocite as having "a good cleavage" and "secondary" as showing "countless irregular cracks." This early-recognized difference in patterns appears to be a real one, and we may conclude that the irregular pattern is formed, rather generally, nearer the surface and at lower temperatures than the regular one. However, the isometric etch pattern, pseudomorphic after bornite, is formed by both supergene and hypogene processes of chalcitization and at all temperatures at which this process takes place.

General Description of Plates

All the figures are photographs of polished sections taken with the reflecting microscope, except Fig. 17, Plate V, which is a photograph of a thin section taken with the polarizing microscope.

The magnifications, which vary from about 10 diameters to nearly 1,000 diameters, should be noted by the reader in order to appreciate the scale on which the described processes are taking place.

Plates I, II, III and IV (except Fig. 13) represent ore from the Bonanza mine, Alaska. They illustrate the occurrence of the early-formed sulphide minerals, represented by a few residual specks; the first generation of chalcocite with complex patterns in blue and white; and the second generation of white chalcocite in veinlets and the accompanying minerals, forming at present at temperatures approaching 0°C.

Plates V, VI and VII (except Fig. 25) show ores from the Apache mines, Pima County, Ariz. Contact metamorphic minerals (garnet) are cut by chalcopyrite and bornite veinlets, and are in turn cut by high-temperature tremolite needles. Hence bornite and first generation chalcopyrite were formed at high temperatures. The complex alterations of bornite are figured, which in this case take place near the surface, just in advance of oxidation. These reactions are governed by the crystallographic directions of the original bornite. The isometric etch pattern of chalcocite which is inherited from bornite is an example of the control of bornite structure on chalcocitization.

Plates VIII, IX and X illustrate the structure of colloidal chalcocite, its porosity and its etch structure. Examples of ordinary orthorhombic etch cleavage are figured so that comparison can be made with the etch structure of meta-colloidal chalcocite and the inherited isometric figures.

²⁰ *Loc. cit.*, p. 57.

PLATE I



FIG. 1.



FIG. 2.

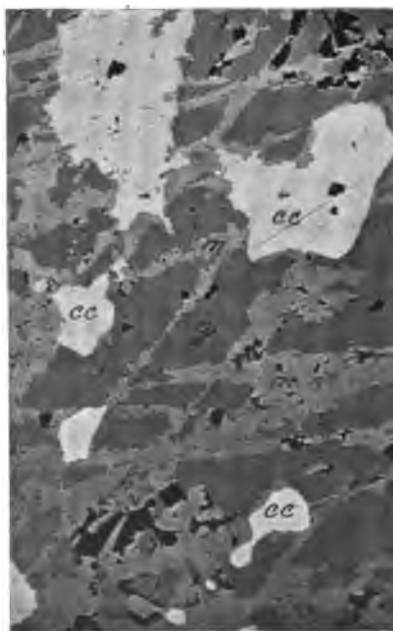


FIG. 3.



FIG. 4.

EXPLANATION TO PLATE I

* FIG. 1.—CALCITE IRREGULARLY REPLACED BY CHALCOCITE.

Klaprothite(?) residual in chalcocite. Second generation of white chalcocite as veinlets in the main mass of first generation of chalcocite.

- cc = 1st generation chalcocite.
 (wcc)₂ = 2d generation chalcocite.
 k = klaprothite(?).
 m = malachite.
 g = calcite gangue.

Magnification, 37 diameters.

FIG. 2.—CHALCOCITE BORDERED BY MALACHITE.

The latter shows selective replacement along one set of the twinning bands in the calcite gangue.

- cc = 1st generation chalcocite.
 cov. = covellite.
 g = calcite gangue.
 m = malachite.

Magnification, 135 diameters.

FIG. 3.—MALACHITE VEINLETS CUTTING THE CALCITE GANGUE, AND BORDERING THE CHALCOCITE.

This relation indicates that the present cold-water circulation, acting especially along the margins of the ore-bodies, is carrying the copper as a carbonate.

- cc = chalcocite.
 m = malachite.
 g = calcite gangue.
 n = holes in surface.

Magnification, 63 diameters.

FIG. 4.—THE LARGEST OF THE SPECKS OF BORNITE RESIDUAL IN CHALCOCITE.

- cc = 1st generation chalcocite.
 dark areas = bornite.

Magnification, 136 diameters.

* Figs. 1-12 and 14-16 inclusive illustrate ore minerals from the Bonanza Mine, Alaska.

PLATE II

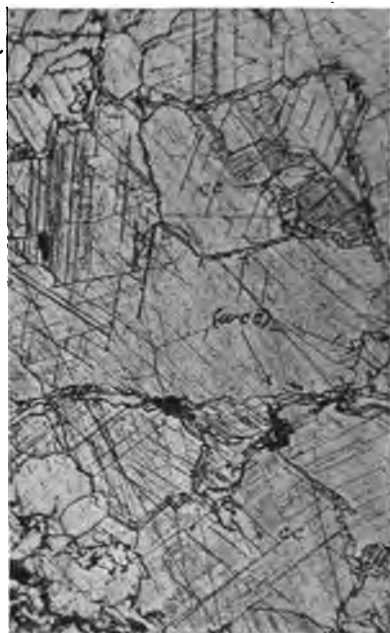


FIG. 5.



FIG. 6.

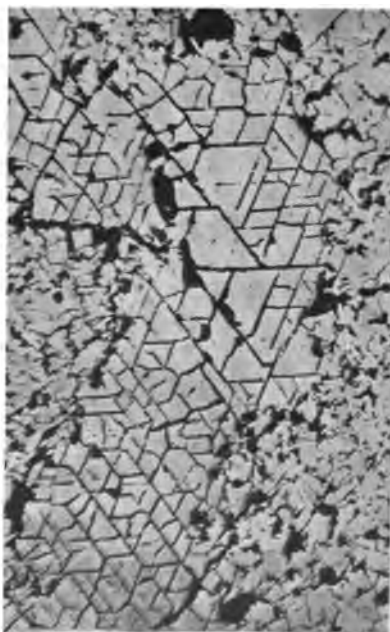


FIG. 7.

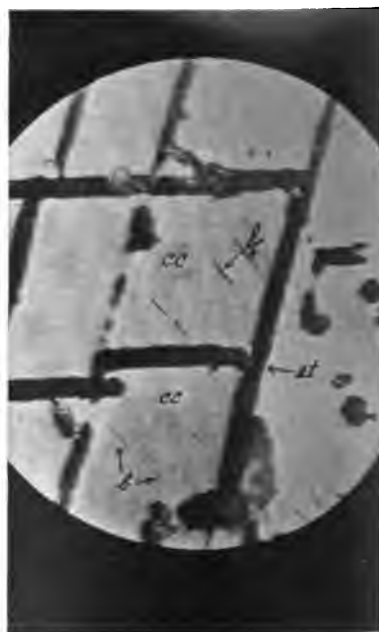


FIG. 8.

EXPLANATION TO PLATE II

FIG. 5.—ORE ETCHED WITH POTASSIUM CYANIDE.

The crystals of chalcocite are distinguished by their regular etched patterns and are bounded by veinlets of second generation of white chalcocite accompanied by crystals of covellite (not shown in photograph—see Fig. 9).

cc = 1st generation chalcocite.

(*wcc*)₂ = 2d generation chalcocite.

Magnification, 55 diameters.

FIG. 6.—BONANZA CHALCOCITE ETCHED BY NITRIC ACID.

Chalcocite of the first generation has a regular triangular etch pattern, and the second generation chalcocite in veinlets etches in one direction only.

Magnification, 51 diameters.

FIG. 7.—BONANZA CHALCOCITE ETCHED BY POTASSIUM CYANIDE.

The large crystal of blue and white chalcocite with a regular triangular etch pattern has been altered along the margins into a white chalcocite with irregular etch pattern containing a mat of covellite needles (not shown in photograph). This is a plain case of unmixing of the first generation chalcocite into white chalcocite and covellite.

Magnification, 14 diameters.

FIG. 8.—BONANZA CHALCOCITE ETCHED BY POTASSIUM CYANIDE.

Two of the three directions of the etch pattern are shown. Stringers of residual bornite are parallel to the third etch direction which is developed elsewhere in the specimen.

cc = chalcocite.

b = bornite.

et = etch lines.

Magnification, 560 diameters.

PLATE III

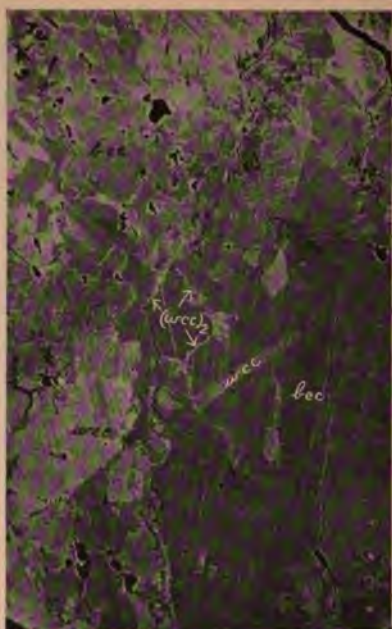


FIG. 9.



FIG. 10.



FIG. 11.

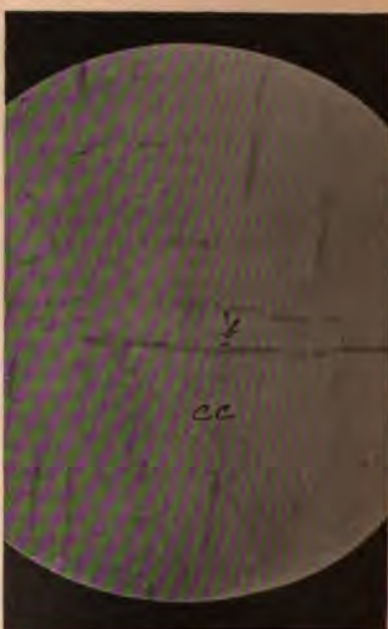


FIG. 12.

EXPLANATION TO PLATE III

FIG. 9.—PATTERN DEVELOPED BY BLUE AND WHITE CHALCOCITE OF THE FIRST GENERATION AND VEINLETS OF WHITE CHALCOCITE OF THE SECOND GENERATION.

To the left are regular areas of white chalcocite pseudomorphic after the "ice cake" structure developed by the irregular replacement of bornite by chalcocite (Fig. 13). On the right the chalcocite develops along regular cleavage lines inherited from bornite. The second generation chalcocite in veinlets and borders around crystals is shown in the center of the photograph accompanied by covellite (black needles).

wcc = white chalcocite, 1st generation.

bcc = blue chalcocite, 1st generation
(*wcc*)₂ = white chalcocite of the 2d generation.

Magnification, 15 diameters.

FIG. 10.—ENLARGEMENT OF A PORTION OF THE FIELD SHOWN IN FIG. 9.

The regular cleavage pattern of white chalcocite, and covellite (black laths) are well shown.

wcc = white chalcocite, 1st generation.

bcc = blue chalcocite, 1st generation
(*wcc*)₂ = white chalcocite, 2d generation.

Magnification, 76 diameters.

FIG. 11.—A COMMON PATTERN IN BLUE AND WHITE CHALCOCITE.

The orthorhombic striation which develops on a much finer scale than the inherited "cleavage," is shown. This development resembles twinning bands.

wcc = white chalcocite, 1st generation.

bcc = blue chalcocite, 1st generation.

Magnification, 436 diameters.

FIG. 12.—RESIDUAL BORNITE STRINGERS IN THE FIRST GENERATION CHALCOCITE.

These stringers are in general parallel to the two-colored patterns and to the cleavage. They are remnants of large crystals of bornite that have been replaced along the cleavage direction of bornite.

cc = chalcocite.

b = bornite.

Magnification, 640 diameters.

PLATE IV

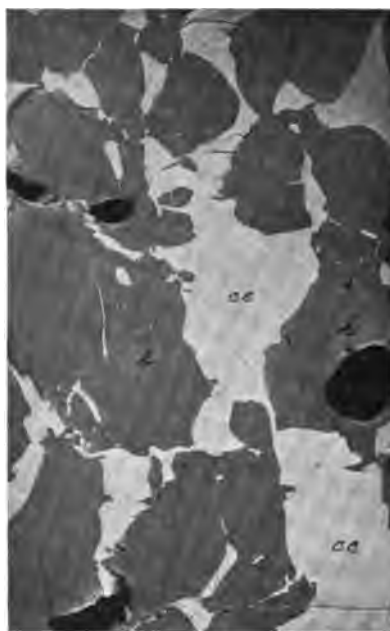


FIG. 13.



FIG. 14.



FIG. 15.

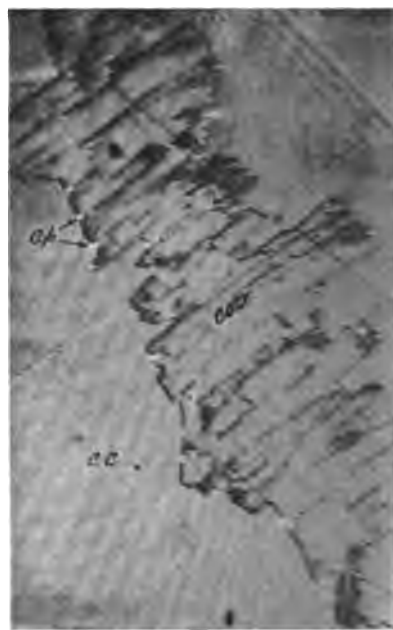


FIG. 16.

EXPLANATION TO PLATE IV

FIG. 13.—TYPICAL "ICE CAKE" STRUCTURE DEVELOPED BY THE FORMATION OF CHALCOCITE ALONG THE MARGINS OF BORNITE CRYSTALS AND NOT ALONG THE INTERIOR CLEAVAGE PLANES.

Ore from Butte, Mont.

cc = chalcocite.

b = bornite.

h = hole.

Magnification, 74 diameters.

FIG. 14.—MALACHITE VEINLETS CUTTING CHALCOCITE AND BORDERED BY ROSETTES OF MELACONITE.

Formed where the ore is oxidizing under present conditions.

cc = chalcocite.

m = malachite.

mel. = melaconite.

Magnification, 13 diameters.

FIG. 15.—COVELLITE NEEDLES DEVELOPING ALONG THE MARGIN OF CHALCOCITE CRYSTALS IN CONNECTION WITH THE SECOND GENERATION OF WHITE CHALCOCITE (NOT TO BE DISTINGUISHED IN THE PHOTOGRAPH).

cc = chalcocite.

cov. = covellite.

h = hole.

Magnification, 46 diameters.

FIG. 16.—AGGREGATES OF COVELLITE NEEDLES GROWING INTO CHALCOCITE WITH CHALCOPYRITE AT THEIR POINTS.

cc = chalcocite.

cov. = covellite.

cp = chalcopyrite.

Magnification, 618 diameters.

PLATE V

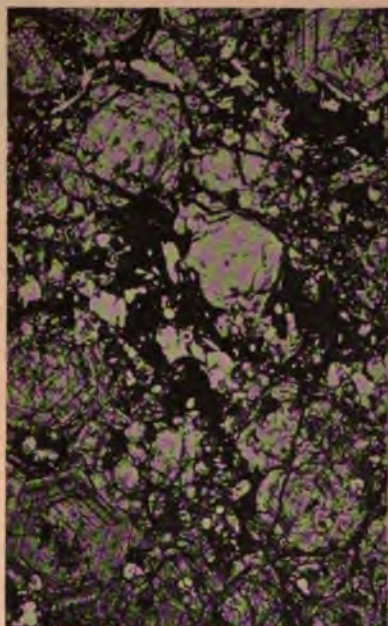


FIG. 17.



FIG. 18.



FIG. 19.

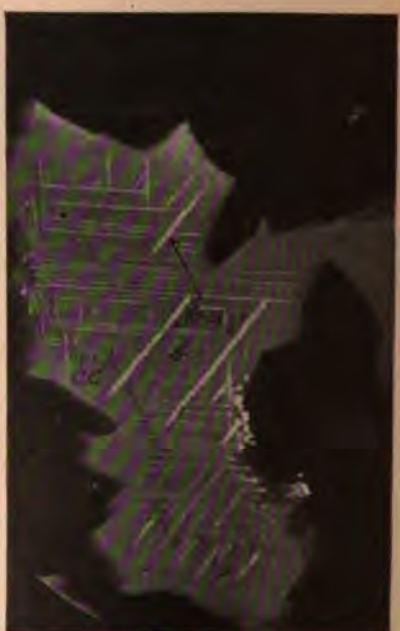


FIG. 20.

EXPLANATION TO PLATE V

* FIG. 17.—GARNET CUT AND REPLACED BY CHALCOPYRITE AND BORNITE.

Gray is garnet. The copper sulphides are black. Thin section, upper nicol out.

Magnification, 17 diameters.

FIG. 18.—CHALCOPYRITE AND BORNITE CUT BY TREMOLITE NEEDLES.

Bornite has been attacked by supergene solutions and developed a second generation of chalcopyrite along the cleavage planes and borders of chalcocite and covellite.

cp = chalcopyrite.

b = bornite.

tr = tremolite.

Magnification, 236 diameters.

FIG. 19.—HIGHER MAGNIFICATION OF A SMALL SPECK OF BORNITE AND CHALCOPYRITE TO SHOW THE SECOND GENERATION CHALCOPYRITE ALONG CLEAVAGE PLANES.

Typical hook-shaped contact developed by the high-temperature bornite and chalcopyrite is shown. Both first generation of chalcopyrite and bornite are bordered by supergene covellite and chalcocite.

cp = 1st generation of chalcopyrite.

(*cp*)₂ = 2d generation of chalcopyrite.

b = bornite.

cc = chalcocite and covellite.

Magnification, 687 diameters.

FIG. 20.—BORNITE WITH CHALCOPYRITE LINES ATTACKED BY CHALCOCITE.

Bornite is less resistant than chalcopyrite to chalcocitization. Hence residual stringers of chalcopyrite are left in the chalcocite.

cc = chalcocite.

¹*b* = bornite.

(*cp*)₂ = 2d generation of chalcopyrite.

Magnification, 720 diameters.

* Figs. 17-24 and 26-28 illustrate ore from Apache Mine, Arizona.

PLATE VI



FIG. 21.

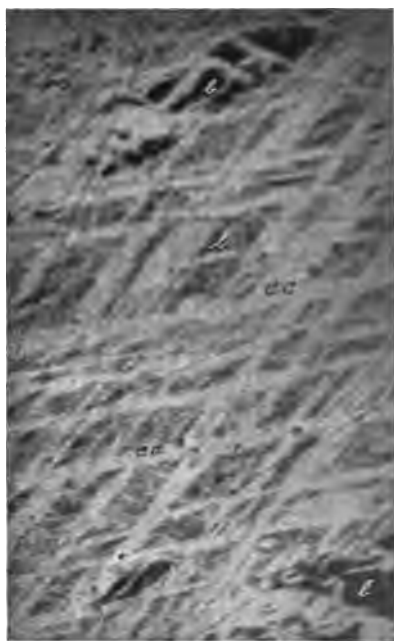


FIG. 22.



FIG. 23.



FIG. 24.

EXPLANATION TO PLATE VI

FIG. 21.—TYPICAL REPLACEMENT OF BORNITE BY CHALCOCITE.

Higher magnification shows the control of bornite cleavage over chalcocitization.

cc = chalcocite.

b = bornite.

tr = tremolite.

Magnification, 55 diameters.

FIG. 22.—BORNITE BREAKING DOWN INTO CHALCOCITE ALONG ITS CLEAVAGES.

Oil was applied to surface immediately after polishing in order to prevent the development of the blue chalcocite pattern.

cc = chalcocite.

b = bornite.

Magnification, 960 diameters.

FIG. 23.—BLUE AND WHITE CHALCOCITE PATTERN PRESERVING BORNITE STRUCTURAL LINES.

A few residual specks of bornite occur in the pattern.

wcc = white chalcocite.

bcc = blue chalcocite.

b = bornite.

Magnification, 800 diameters.

FIG. 24.—CHALCOPYRITE REPLACING BORNITE ALONG STRUCTURAL LINES WITH CHALCOCITE EATING INTO CHALCOPYRITE ALONG THE SAME PLANES.

This structure was not noted in chalcopyrite of the first generation.

cc = chalcocite.

b = bornite.

cp = chalcopyrite.

Magnification, 800 diameters.

PLATE VII



FIG. 25.



FIG. 26.



FIG. 27.



FIG. 28.

EXPLANATION TO PLATE VII

FIG. 25.—ORE FROM ENGEL'S MINE, SHASTA COUNTY, CALIFORNIA. BORNITE REPLACED BY COVELLITE ALONG CRYSTAL PLANES OF THE FORMER.

In this case chalcocite replaces the bornite irregularly and is not governed by the bornite structure. Photo by A. F. Rogers.

cc = chalcocite.

b = bornite.

cov. = covellite.

Dark areas = silicates.

Magnification, 364 diameters.

FIG. 26.—MALACHITE REPLACING CHALCOCITE ALONG PLANES INHERITED FROM BORNITE CLEAVAGE.

The blue and white patterns in the chalcocite show that the crystals of chalcocite are not parallel to the bornite cleavages and that we have here a structural pseudomorphism and not crystalline pseudomorphism. This is believed to be the case generally.

bcc = blue chalcocite.

wcc = white chalcocite.

m = malachite.

Magnification, 960 diameters.

FIG. 27.—ISOMETRIC ETCH PATTERN IN CHALCOCITE INHERITED FROM BORNITE.

An area of blue and white chalcocite similar to that shown in Fig. 23, etched with HNO_3 .

Magnification, 800 diameters.

FIG. 28.—MARGIN OF PATTERN SHOWN IN FIG. 27.

Outside the blue and white pattern, the inherited bornite structure does not control.

Magnification, 960 diameters.

NOTE.—Figs. 23, 24, 26, 27 and 28 are taken from one speck of chalcocite about $\frac{1}{16}$ in. in diameter.

PLATE VIII



FIG. 29.



FIG. 30.

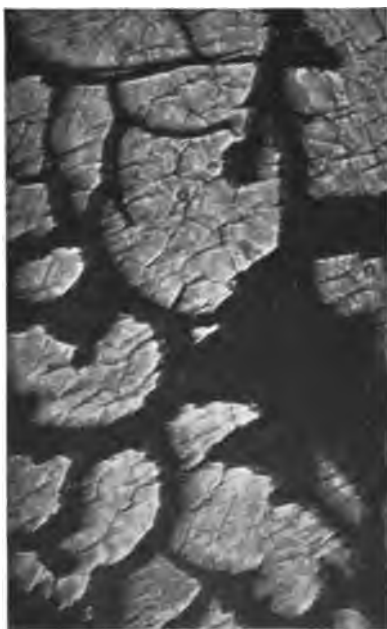


FIG. 31.

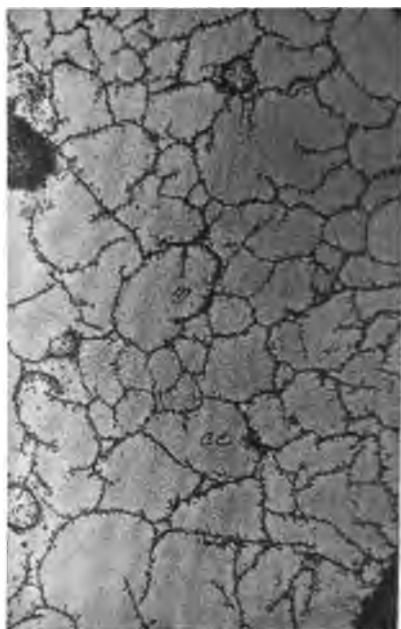


FIG. 32.

EXPLANATION TO PLATE VIII

FIG. 29.—ORE FROM BINGHAM CANYON, UTAH, COLLECTED BY A. F. ROGERS.

Meta-colloidal concentric structure is brought out by weathering.

Light areas = chalcocite.

Darker areas = copper silicates and carbonates.

Magnification, 14 diameters.

FIG. 30.—A PORTION OF ONE RING OF THE PATTERN SHOWN IN FIG. 29.

Magnification, 44 diameters.

cc = chalcocite.

m = malachite.

FIG. 31.—ETCHED AREA ON SPECIMEN SHOWN IN FIGS. 29 AND 30.

This shows the same structure in finer detail.

Magnification, 314 diameters.

FIG. 32.—META-COLLOIDAL CHALCOCITE, LOCALITY UNKNOWN.

Porosity of the chalcocite is shown in the photograph by its grayish speckled appearance. Branching veinlets are of malachite.

cc = chalcocite.

m = malachite.

Magnification, 11 diameters.

PLATE IX

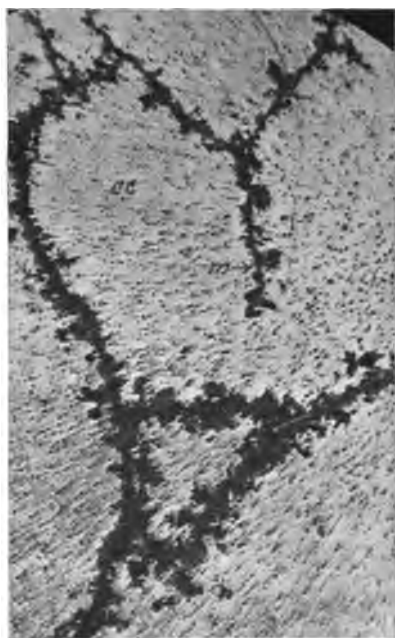


FIG. 33.

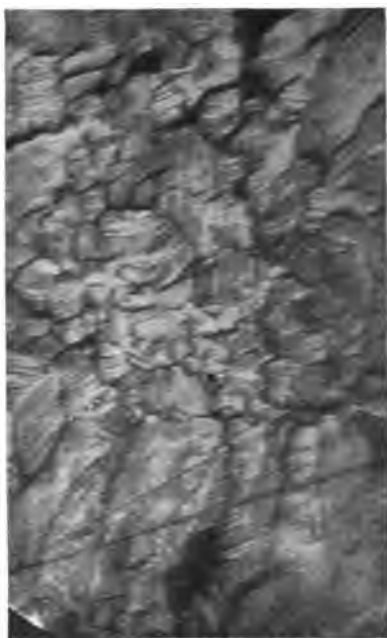


FIG. 34.

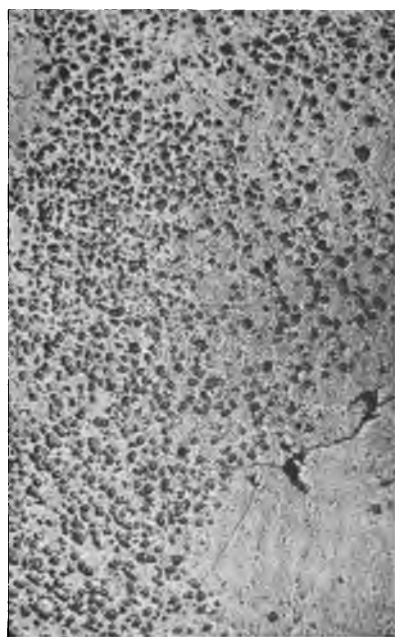


FIG. 35.

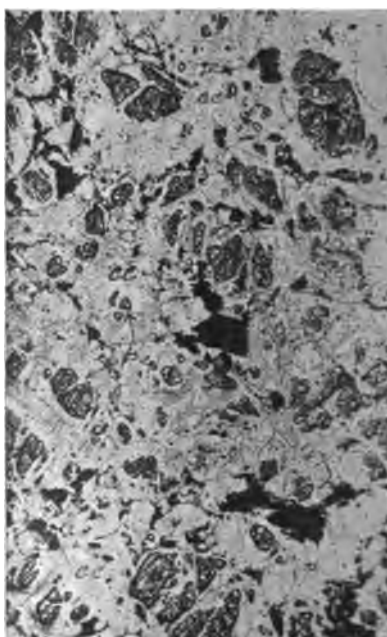


FIG. 36.

EXPLANATION TO PLATE IX

FIG. 33.—HIGHER MAGNIFICATION OF FIG. 32, TAKEN IN ORDER TO SHOW THE POROSITY OF THIS TYPE OF CHALCOCITE.

Magnification, 93 diameters.

FIG. 34.—ETCH STRUCTURE (HNO_3) DEVELOPED IN THE SPECIMEN FIGURED IN PHOTOGRAPHS 31, 32 AND 33.

Each fine grain shows a striation in one direction. A similar pattern was developed on all chalcocites here figured as meta-colloidal.

Magnification, 960 diameters.

FIG. 35.—NODULAR ORE OF THE RED BEDS TYPE (?), LOCALITY UNKNOWN.

The chalcocite cements very fine sand and is exceedingly porous. Some idea of the size of the pores can be obtained by comparing the pattern in gray specks with the sand grains.

Magnification, 11 diameters.

FIG. 36.—“EXPLODING BOMB,” STRUCTURE IN ORE FROM THE TOP OF THE SULPHIDE ZONE AT MIAMI, ARIZ.

Pyrite is forced open by the deposit of colloidal chalcocite. Streaks of porous chalcocite (dappled in photograph) outline the original pyrite crystals.

cc = chalcocite.

p = pyrite.

h = hole.

Magnification, 11 diameters.

PLATE X

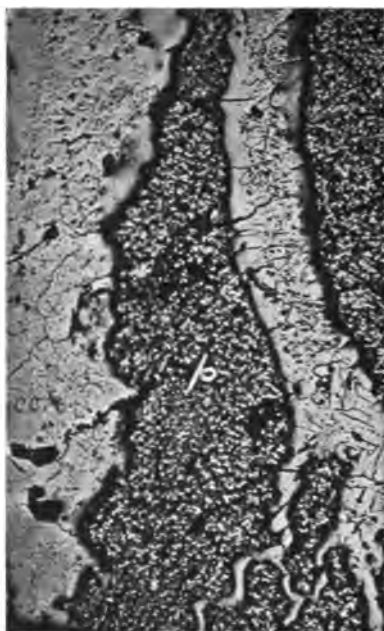


FIG. 37.]

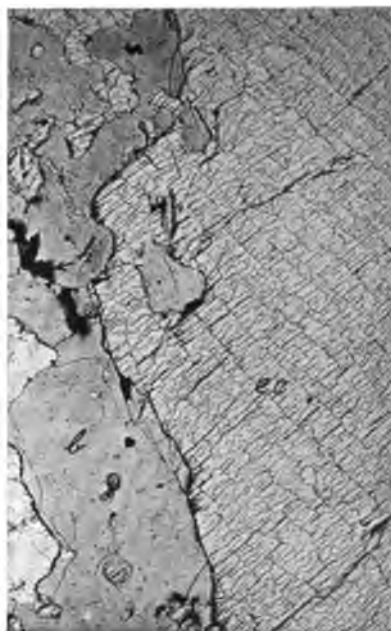


FIG. 38.

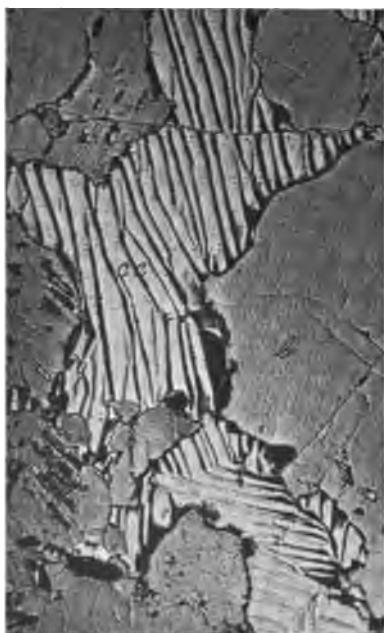


FIG. 39.

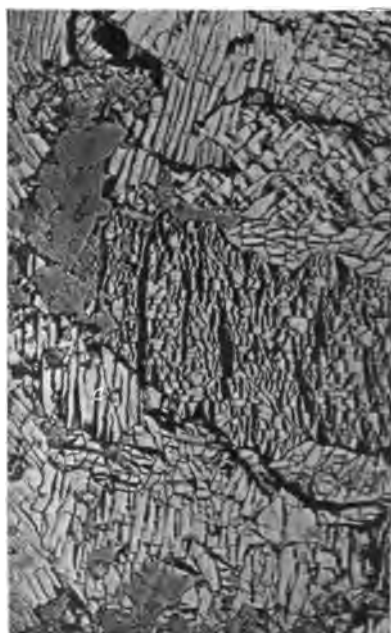


FIG. 40.

EXPLANATION TO PLATE X

FIG. 37.—MAGNIFICATION OF A VEINLET OF CHALCOCITE REPLACING PYRITE FROM SPECIMEN SHOWN IN FIG. 36.

The vein develops an imperfect conchoidal parting and considerable porosity in the central portion (not etched).

Magnification, 59 diameters.

FIG. 38.—TYPICAL ORTHORHOMBIC ETCH CLEAVAGE IN CHALCOCITE.

This cleavage shows one direction of parting more distinct than the other direction. From Red Wing Mine, Virgilina District, Virg. The ore shows the so-called "graphic intergrowth" and is believed by Dr. Laney to be formed by ascending solution.

cc = chalcocite.

b = bornite.

Magnification, 56 diameters.

FIG. 39.—TYPICAL ORTHORHOMBIC ETCH CLEAVAGE IN BUTTE CHALCOCITE CONSIDERED AS HYPOGENE BY SALES, ROGERS, RAY AND THOMPSON.

From the same specimen as shown in Fig. 13. The cleavage or parting is pronounced in one direction:

cc = chalcocite.

b = bornite.

Magnification, 74 diameters.

FIG. 40.—FROM THE SAME SPECIMEN AS FIG. 39.

A parting in two directions is noted in the center of the field but is more regular in one direction than in the other.

cc = chalcocite.

b = bornite.

Magnification, 74 diameters.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Grinding Brass Ashes in the Conical Ball Mill

BY ARTHUR P. TAGGART* AND R. W. YOUNG,* NEW HAVEN, CONN.

(New York Meeting, February, 1916)

FOREWORD

THE tests herein described are part of an extended series of experiments, performed by the authors together with J. F. McClelland and L. W. Bahney, on the reclamation of metallics from foundry and manufacturing-plant scrap. The most obstinate problem encountered was the proper cleaning of the metal.

INTRODUCTION

In the manufacture of brass and brass products the weight of the finished product leaving the plant is much less than the total weight of copper and zinc entering. A small part of this discrepancy is due to the unavoidable oxidation of the metals in melting, but by far the largest part of the loss is in the form of metallic particles of copper, zinc, and brass. These losses occur principally at three points in the process: (a) in the casting-shop ashes; (b) in the slags from furnaces melting scrap brass; (c) in material spilled on the floor throughout the manufacturing process. The amount of these products in some plants is nearly 100 tons per 24 hours.

Casting-shop ashes contain the metal spilled from the crucibles during melting and pouring, and the metal included in the slags formed in the melting process. The ashes also contain pieces of broken crucibles with adhering metal; 20 to 30 per cent. of unburned coal is also present. The metal in casting-shop ashes contains clean pieces of zinc or copper ingots weighing 1 to 2 lb., jagged pieces of metal of all shapes ranging in size from several inches down to the finest dust, and shot and botryoidal shapes ranging in size from 2 or 3 in. down to 0.01 mm. or less in diameter.

Slags from furnaces melting scrap brass contain metal in the form of small shot.

Floor sweepings occur in all shapes: spirals, thin shavings, pins and pin-like pieces, wire, rods, and buffing-wheel dust are the commonest.

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In general the problem of reclaiming this metal has been to free the metal from adhering slag and separate it from the associated slag, coal and cinders. On its face this problem appears simple. That it is not simple, however, is attested by the concentrations of fine metallic particles in the sewers and stream beds of the New England brass-manufacturing districts. The usual plant installed has consisted of a crude hand-picking installation for recovering the coarse metal; a restricted-discharge grinder, such as the Huntington or Chilian mill, Krupp ball mill, Hill cinder crusher, etc., for severing the metal from the slag; a concentrating table, usually a Wilfley, for treating the fine discharge from the grinder; and a settling tank for the slimes flowing away from the concentrating table. The coarse metallics are collected intermittently by stopping the plant and cleaning out the grinding machine. These plants are claimed to throw away tailings assaying but 0.5 per cent. copper. Such statements are based on erroneous samples, for the tailings will, when the plant is properly sampled, be found to contain up to 5 per cent. copper, and most of the coal is wasted.

In connection with the design of a plant for the treatment of brass ashes the writers were confronted with the problem of finding a grinder that would answer the following requirements: (a) take a feed containing particles of metal 2 or 3 in. in diameter, with adhering slag, and at the same time pieces of slag containing metal shot in all sizes from the above down to the finest; (b) discharge continuously both the metallic and non-metallic constituents of the feed, and thus eliminate the intermittent factor attendant upon the operation of the grinders previously mentioned; (c) discharge a product of such character that all the material larger than 2.5 mm. is clean metal, while all the slag and cinders are ground to pass the same screen; (d) have a capacity of about 1 ton per hour; (e) have a reasonable repair and power charge. The following are the results of two tests run on a 4½-ft. by 16-in. conical ball mill with these requirements in mind.

Description of the Test

The mill used in these tests was a special one built by the Hardinge Conical Mill Co. for testing purposes and installed in the Hammond Mining and Metallurgical Laboratory, Sheffield Scientific School, Yale University. As will be seen from Fig. 1, the mill is mounted with the discharge end *a* on a fixed foundation *b*. The feed end *c* is carried on the wooden pier *d* which can be slid parallel to the long axis of the mill and fastened in any desired position by holding-down bolts in the floor slots *e*. The mill can be broken along the rims *f* and the cylinder *g* removed, thus giving a mill with no cylindrical section, or one, two, or three cylinders can be inserted making the cylindrical portion of the mill 16 in., 32 in., and 48 in. long respectively as desired. The mill is lined with chrome-

steel lifting bars. For the tests described herein, the following are the mill adjustments:

One 16-in. cylinder used, making the mill the standard $4\frac{1}{2}$ ft. by 16 in.; tilt, 2.5 in. toward the discharge end; speed, 28 to 29 r.p.m.

Ball Charge:

Size of Balls, Inches	Number of Balls	Weight, Pounds
5	100	1,927
4	100	951
3	205	815
$1\frac{3}{4}$	1,002	720
Total		4,413

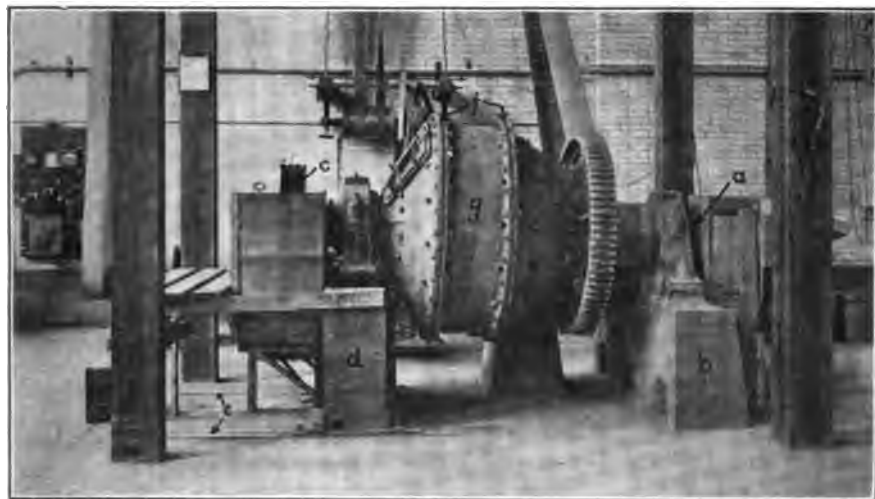


FIG. 1.

Details of Tests.—Two runs were made, which are referred to as No. 1 and No. 2 in the following notes.

Run No. 1.—Feed, sluice concentrate, containing 30.01 per cent. coal. (For sizing test see Table I.)

Feed rate, 2.24 tons solid per hour.

Moisture in feed, 42.9 per cent.

Power consumption, 20 hp.

Run No. 2.—Feed, first- and second-gate discharges from jig, containing 1.23 per cent. coal. (For sizing test see Table III.)

Feed rate, 0.91 tons solid per hour.

Moisture in feed, 39.7 per cent.

Power consumption, 21 hp.

Products.—Sizing-sorting-assay tests of feed and product of the two runs are given in Tables, I, II, III, and IV.

Summary of Results

1. A concentrate can be made by simply screening the discharge from the mill. The grade of this concentrate will depend on the character of the feed to the mill and the aperture of the screen used. Since brass containing more than 10 per cent. "dirt" (slag and coal) is unsuitable for remelting, the concentrate obtained by screening the product from Run No. 1 is not sufficiently clean. The product from Run No. 2 can be screened on a 1.651-mm. screen and will give a concentrate containing 90.9 per cent. clean metal. If cleaner metal is desired, it can be obtained by using a screen of larger aperture. (See Tables II and IV.)

2. The material passing the 1.651-mm. screen in Run No. 2 (see Table IV) containing 13.9 per cent. metal, is too rich to be thrown away, as is done in some plants, or to be sold to a junk dealer at a couple of dollars a load, as is done in some others.

3. In Run No. 2 about 25 per cent. of the total feed to the mill, containing 70 per cent. of the total metallic in the feed, can be saved clean enough for remelting without the use of concentrating machines. (See Table IV.)

TABLE I.—*Sizing-Sorting Test on Feed. Run No. 1*

On Screen		Tons per 100 Tons	Cumula- tive Per Cent.	Assay, Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, millimeters	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal	
18.85	670.00	1.14	1.14	20.5	0.285	0.335	0.090	0.520	25.0
13.33	4,350.00	7.87	9.01	20.8	1.516	3.680	1.104	2.674	20.1
9.423	10,260.00	18.60	27.61	11.4	3.255	9.710	0.168	5.635	20.1
6.680	10,510.00	19.05	46.66	17.7	5.229	8.592	0.176	5.229	22.0
4.699	7,295.39	13.22	59.88	23.9	4.100	5.950	0.952	3.170	24.0
3.327	7,681.80	13.92	73.80	17.2	3.638	6.840	0.197	3.442	24.4
2.362	5,081.21	9.21	83.01	17.0	1.576	5.060	0.926	2.574	23.6
1.651	2,655.15	4.81	87.82	13.2	0.529	2.763	0.485	1.518	22.9
1.168	1,517.14	2.75	90.57	10.4	0.254	1.135	0.202	1.361	22.5
0.833	1,223.11	2.21	92.78	6.2	0.103	0.749	0.117	1.358	22.1
0.589	1,107.47	2.00	94.78	4.8	0.107	1.125	0.047	0.768	21.7
0.417	780.74	1.41	96.19	8.6	0.147	0.710	0.047	0.553	21.6
0.295	641.08	1.16	97.35	8.3	0.081	0.621	0.072	0.458	21.4
0.208	433.23	0.78	98.13	8.8	0.064	0.421	0.045	0.295	21.3
0.147	326.12	0.59	98.72	4.7	0.027	0.367	0.017	0.196	21.2
0.104	289.23	0.52	99.24	8.9	0.061	0.295	0.013	0.164	21.1
0.074	125.09	0.23	99.47	9.8	0.026	0.144	0.010	0.060	21.1
Through	0.074	0.53	100.00	10.6	0.085	0.354	0.005	0.091	21.1
Loss	80.29								
Totals	55,320.00	100.00			21.083	48.851	4.673	30.066	

TABLE II.—*Sizing-Sorting Test on Product. Run No. 1*

On Screen		Tons per 100 Tons	Cumula- tive Per Cent.	Assay, Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen	
Aperture, millimeters	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal		
13.33	15.12	0.240	0.240	17.5	0.067	0.023	0.000	0.150	27.9	
9.423	62.80	0.995	1.235	44.6	0.692	0.060	0.018	0.243	61.4	
6.680	211.78	3.360	4.595	44.9	2.306	0.449	0.104	0.605	66.7	
4.699	326.68	5.180	9.755	48.3	3.942	0.539	0.061	0.699	71.6	
3.327	401.79	6.368	16.143	42.6	4.038	1.036	0.298	1.294	68.4	
2.362	436.05	6.912	23.055	41.2	2.829	2.232	0.400	1.851	60.2	
1.651	425.68	6.752	29.807	25.7	1.437	3.672	1.335	1.643	51.4	
1.168	441.61	7.003	36.810	12.5	0.662	2.956	0.733	3.384	43.4	
0.833	498.17	7.900	44.710	10.5	0.313	5.083	1.008	2.504	36.4	
0.589	550.18	8.725	53.435	8.1	0.488	4.923	0.637	3.314	31.4	
0.417	563.52	8.939	62.374	8.0	0.635	4.780	0.503	3.524	27.9	
0.295	481.63	7.641	70.015	6.6	0.342	4.396	0.471	2.903	25.4	
0.208	385.10	6.105	76.120	7.3	0.154	3.772	0.559	2.180	23.5	
0.147	362.65	5.754	81.874	6.8	0.091	3.796	0.535	1.867	22.0	
0.104	328.29	5.207	87.081	6.5	0.078	3.571	0.467	1.558	20.7	
0.074	98.28	1.560	88.641	6.7	0.008	1.374	0.159	0.178	20.4	
Through Loss	0.074	715.80	11.359	100.000	9.3	0.011	8.829	1.672	2.519	18.1
Totals	6,410.00	100.000			18.093	51.491	8.960	30.416		

TABLE III.—*Sizing-Sorting Test on Feed. Run No. 2*

On Screen		Tons per 100 Tons	Cumula- tive Per Cent.	Assay Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen	
Aperture, Mm.	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal		
26.67	323.00	7.810	7.810	55.9	6.285	1.525	0.698		80.5	
18.85	363.0	8.775	16.585	29.2	1.524	7.251	2.588		47.1	
13.33	1,065.00	25.757	42.342	25.8	6.045	19.591	5.425	0.121	32.7	
9.423	1,088.00	26.307	68.649	31.6	8.340	17.846	4.970	0.121	32.6	
6.680	720.00	17.420	86.069	36.1	7.257	9.800	2.808	0.363	34.2	
4.699	380.00	9.189	95.258	42.9	5.200	3.747	1.115	0.242	34.6	
3.327	91.33	2.210	97.468	43.5	1.219	0.896	0.320	0.095	36.8	
2.362	28.16	0.681	98.149	43.9	0.380	0.267	0.098	0.034	36.2	
1.651	19.91	0.482	98.631	37.6	0.221	0.246	0.069	0.015	37.0	
1.168	9.90	0.239	98.870	32.7	0.098	0.125	0.026	0.016	37.0	
0.833	6.27	0.152	99.022	25.7	0.033	0.102	0.030	0.017	37.0	
0.589	6.00	0.145	99.167	21.4	0.025	0.096	0.024	0.024	37.0	
0.417	4.58	0.111	99.278	17.1	0.018	0.068	0.013	0.025	36.9	
0.295	4.86	0.118	99.396	15.3	0.015	0.077	0.013	0.026	36.9	
0.208	4.63	0.112	99.508	10.7	0.001	0.084	0.018	0.027	36.8	
0.147	4.73	0.114	99.622	10.5	0.002	0.079	0.018	0.033	36.8	
0.104	5.00	0.121	99.743	7.4	0.001	0.084	0.014	0.036	36.8	
0.074	2.73	0.066	99.809	9.1	0.001	0.051	0.008	0.014	36.7	
Through Totals	0.074	7.91	0.191	100.000	13.1	0.010	0.161	0.029	0.020	36.7
	4,135.01	100.000			36.675	62.096	18.324	1.229		

TABLE IV.—*Sizing-Sorting Test on Product. Run No. 2*

On Screen		Tons per 100 Tons	Cumula- tive Per Cent.	Assay Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, Mm.	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal	
9.423	79.00	1.316	1.316	62.5	1.316	0.000	0.000	0.000	100.0
6.680	233.00	3.881	5.197	61.8	3.827	0.054	0.014	0.000	99.0
4.699	374.00	6.229	11.426	61.3	6.075	0.138	0.040	0.016	98.3
3.327	308.50	5.138	16.564	56.4	4.382	0.703	0.248	0.053	94.3
2.362	328.50	5.475	22.039	57.7	4.850	0.592	0.205	0.033	92.9
1.651	248.00	4.131	26.170	53.1	3.310	0.686	0.197	0.135	90.9
1.168	183.90	3.063	29.233	45.8	2.090	0.875	0.168	0.108	88.4
0.833	175.90	2.930	32.163	36.3	1.399	1.387	0.302	0.144	84.7
0.589	243.90	4.066	36.229	20.2	0.960	2.721	0.352	0.385	77.8
0.417	308.00	5.132	41.361	12.5	0.553	3.837	0.477	0.742	69.5
0.295	428.10	7.135	48.496	9.3	0.215	6.024	0.841	0.896	59.7
0.208	432.80	7.211	55.707	8.6	0.213	5.880	0.784	1.118	52.4
0.147	460.80	7.680	63.387	10.7	0.233	6.338	1.085	1.109	46.1
0.104	507.00	8.446	71.833	9.9	0.183	7.222	1.163	1.041	41.4
0.074	299.30	4.987	76.820	11.5	0.161	4.262	0.756	0.564	38.5
Through	0.074	1,393.20	23.180	100.000	11.5	4.265	17.067	1.848	34.0
Loss		206.10							
Totals		6,210.00	100.000		34.022	57.786	6.632	8.192	

TABLE V.—*Elutriation Test on Minus 0.074-mm. Material, Product of Run No. 2*

Current Mm. per Sec.	Weight, Grams	Tons Per 100 Tons Original Product	Per Cent.	Cumula- tive Per Cent.	Assay Cu, Per Cent.	Tons Per 100 Tons of Original Product		
						Metallic	Slag	Coal
9.5	0.45	0.241	1.09	1.09	51.1	0.197	0.044	0.000
5.1	2.80	1.499	6.80	7.89	23.5	0.563	0.936	0.000
4.2	1.67	0.896	4.06	11.95	9.0	0.130	0.766	0.000
3.2	4.25	2.273	10.32	22.27	12.1	0.440	1.833	0.000
2.3	3.65	1.955	8.87	31.14	10.0	0.313	1.617	0.025
1.5	3.87	2.071	9.39	40.53	9.6	0.317	1.547	0.207
0.9	4.99	2.672	12.13	52.66	9.8	0.419	1.932	0.321
0.6*	9.09	4.869	22.09	74.75	9.9	0.771	3.528	0.570
A	10.12	5.435	24.67	99.42	10.4	0.905	3.929	0.601
B	0.24	0.128	0.58	100.00	0.0	0.000	0.095	0.033
Totals	41.13	22.039	100.00			4.055	16.227	1.757

* The discharge from the elutriation apparatus with 0.6 mm. per second current was led into the bottom of a 2-liter wide-mouth bottle. A siphon dipping about 1 in. into the neck of this bottle carried the fine slime to other bottles rigged up in the same way. After standing several hours the supernatant liquid in these bottles was decanted and evaporated to dryness giving the product marked B; the material which settled in all but the first bottle is marked A. The material which settled in the first bottle is that opposite the 0.6 mm. per second current.

4. Comparison of the sizing tests on feed and product of Runs Nos. 1 and 2 shows that finer grinding was done in Run No. 2. This is due partly to the lower rate of feed and partly to the large percentage of coal in the feed in Run No. 1. It will be noted (Table II) that more than 50 per cent. of the non-metallic material remaining on the 2.362-mm. screen is coal. A comparison of results shows that the coal does not grind as freely as the slag.

5. Analysis of the data given shows that the metallics have been reduced in size (6.08 mm. to 3.62 mm. average size in Run No. 1, and 11.83 mm. to 3.01 mm. average size in Run No. 2). The reduction is not, however, so great as indicated, by reason of the fact that the largest pieces of metal were held back in the mill, the duration of the runs not being great enough for them to work through. The writers believe that the sharp increase in the percentage of minus 0.074-mm. metallics in Run No. 2 is due, not to sliming, but to the freeing of the included metallic from the slag, as was not done in Run No. 1. This belief was borne out by an elutriation test (Table V) which showed no metallic in the finest sizes, and was confirmed by microscopic examination of the minus 0.074-mm. material, which revealed the majority of the free metallics to be rounded, and not jagged as would be expected had they been abraded from the larger pieces.

6. The tables show the reduction in size of the slag in the two runs and the necessity of this reduction in order to free the metal. Microscopic examination of a slag particle, 0.1 mm. in diameter revealed the presence of metal shot not more than 0.01 mm. in diameter. In order to free these shot for concentration the greatest possible reduction of the slag is essential.

7. The reduction in size of coal is not so great in Run No. 1 as in Run No. 2. In both cases, however, the bulk of the coal will pass a 1-mm. screen, and is, therefore, too fine for burning economically without special appliances. Its value to the plant is much less than if it were removed from the ashes before grinding.

Conclusions

1. The 4½-ft. by 16-in. conical ball mill, operated as described in Run No. 2, if fed a dirty brass concentrate containing slag and a little coal, at the rate of 1 ton per hour, will grind the slag and coal so that a great part of it will pass a 1.651-mm. screen while the brass in the feed will be only slightly reduced in size and can be separated from the dirt by screening. The screening operation can be performed by attaching a trommel of the proper aperture to the discharge end of the mill.

2. If the feed to the mill contains a considerable percentage of coal, the product saved on the screen will contain more than 10 per cent.

"dirt." The writers believe that this will be true even if the feed rate is considerably below 1 ton per hour.

3. As compared with the other types of machine at present being used for grinding brass ashes, the conical ball mill should show a decreased repair cost, since screens are eliminated, and it has a further advantage in that it discharges continuously not only the ground slag and coal but also the cleaned metal.

4. The metal is not slimed in the mill, but that which is not caught clean on the screen is of such size that it can be easily saved by proper table treatment.

5. While the basis for the conclusion does not appear from these tests, the writers have found in other work along similar lines that with the mill horizontal the metal is not cleaned up and the large pieces of metal will not discharge.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Modern Development in the Combustion of Blast-Furnace Gas with Special Reference to the Bradshaw Gas Burner

BY K. HUESSENER,* PITTSBURGH, PA.

(New York Meeting, February, 1916)

INTRODUCTION

THIS paper attempts a survey of the principles involved in the combustion of blast-furnace gas in boilers and stoves. I do not expect to be able to give much information which is actually new, since the laws and methods of calculation pertaining to this problem are contained in a number of handbooks; but, so far as I am aware, they have nowhere been treated exclusively with reference to this one problem of gas economy. The road which I follow is less that of strictly scientific research than that of the practical engineer, faced by operation problems which require a quick solution by easily accessible means without a prolonged search in scientific handbooks. For this purpose I give a number of tables and diagrams, as well as the results of many years' practical working experience as combustion engineer, which I trust will prove useful to my readers.

HISTORICAL

Considering the tremendous efforts made during the last 15 years to increase the efficiency of all kinds of power plants, it is highly surprising that until a comparatively very recent period, progress in the economical combustion of blast-furnace gas has been so slow.

The explanation generally put forth, that fuel has been so cheap that it was hardly worth while to bother about this economy, will not hold good, for the greatest trouble was taken to improve the combustion of coal and to reduce the consumption of steam. The most plausible explanation appears to be the general impression, held as an accepted fact, that gas, especially hot, uncleaned, blast-furnace gas, was not a good boiler fuel. Many managers of high repute had tried unsuccessfully to improve the conditions of its use, and the dictum went forth that efficiencies of over 60 per cent., particularly with high loads, were impossible. This was the situation not only in the United States, with very cheap fuel, but also in Europe, where fuel prices are twice and three times as high.

* President, The Huessener Engineering Co., Inc.

The failure to improve conditions was, to a large extent, due to the type of boilers mostly in use up to the last five or six years. This also is true for both America and Europe. In America, one-pass boilers of the Cahall, Cook and Wheeler types were used in large numbers, and the boiler capacity was not always sufficiently large so that overload was frequently necessary. While with up-to-date combustion arrangements fair efficiencies can be obtained at the present time from these one-pass boilers, they have baffled at least all my efforts to secure that result with an overload.

In Europe 90 per cent. of all blast-furnace boilers were internal-flue horizontal boilers, and as the gas was not cleaned, dirt accumulated in the flues and rendered good efficiencies impossible, however good the combustion itself might have been. The natural expedient in this case was to clean the gas.

In the case of water-tube boilers it was of course possible to remove the dust by the use of soot blowers or steam- or air-blowing lances. This could not be done with the internal-flue boiler, where in order to remove the dust it was necessary to lay off the boilers altogether. With this type of boiler both efficiencies and loads declined continuously, and after one week's working it was impossible to run a boiler on rated load. The natural expedient was to clean the gas; and accordingly in the beginning of the century a number of efficient gas-cleaning plants were developed in Germany. The cleaning of the gas rendered the use of blast-furnace gas in gas engines possible. At a time when cleaning plants, which formed so large an item in the total expenditure over a gas-engine installation, were anyhow considered indispensable and furthermore when an efficiency of 50 per cent. on the gas-fired boilers was considered the best possible obtainable (as was the case 15 years ago), the superiority of the gas engine was hardly open to discussion. In this connection it must be kept in mind that during the first years of gas engines, it was generally accepted that they would not consume more than 14,000 B.t.u. per kilowatt. At this time the best steam engines would still consume 20 lb. of steam per horsepower, equivalent to 26,000 B.t.u. per kilowatt. If then a kilowatt had to be raised at an efficiency of 50 per cent., it meant a consumption of 52,000 B.t.u. for the steam engine, as compared with 14,000 B.t.u. for the gas engine. Since most people considered the gas-fired boiler at that time a thing of the past, little effort was spent for a number of years in improving the combustion arrangement. This development was also held back because the gas-cleaning plant by itself brought about, with the internal-flue boiler, a marked improvement both in efficiency and load, so that an efficient combustion was no longer the all-important factor.

After the gas engines had been in use at a number of places, however, they were found not to be an unqualified blessing; they gave high

efficiencies only with high and constant loads, which in iron and steel works loads the blowing engines can furnish, but which could not be maintained in mill work. In 1907, Mr. Hoff, a German iron-works manager of high repute, investigated 32 power plants using gas engines at various German steel works and ascertained that the average load of all of them was only 52 per cent., that, in one instance, it was as low as 32 per cent.; and that it had not been possible in any case to generate a kilowatt with the low consumption of 14,000 B.t.u., a claim put forth by the makers of gas engines.

Today it is generally accepted that, even at so-called "mixed" power plants, where only the constant load is carried by the gas engines, and the excess load by steam turbines, 18,000 B.t.u. are required for each kilowatt. Moreover, since for gas engines the gas has to be cooled down to atmospheric temperature, we must add to the 18,000 B.t.u. actually consumed the sensible heat which has been taken out of the gas in the washing process; and as this sensible heat represents 8 per cent. of the latent heat, the actual heat consumption of the gas engine under the most favorable conditions stands today at 19,400 B.t.u. per kilowatt. In this country, the General Electric Co. claims that with its steam turbine it can raise a kilowatt with an actual heat expenditure of 19,000 B.t.u.

During this development of the gas engine, great strides were made in the improvement of the steam engine by the advent and development of the steam turbine. The 20 lb. of steam formerly required per horsepower, was gradually reduced to 14 lb. and, in some cases, where very large superheats were used (in Germany one is going as far as 450°F.) to 11 lb. per horsepower.

This concurrent development of steam engine and steam turbine brought about a revival of interest in the perfection of combustion. In the meantime, the waste-heat coke oven, which, up to about 1900, was almost universally used in Germany, gave way to the regenerative coke oven, but this development was generally retarded by the conviction everywhere held that gas was not suitable for boiler firing, and that, unless it could be used in gas engines or sold to gas works, the adoption of the regenerative coke oven would always mean a sacrifice in steam.

The comparative figures in this respect put forth as recently as 1912 by Baron Coppée at the British Iron and Steel Institute Meeting in Brussels were 0.75 ton of steam per ton of coal from the regenerative oven, as against 1 ton of steam per ton of coal from the waste-heat oven.

BURNERS

It is a remarkable fact that in Germany the first efficient gas burners were developed for coke-oven gas, although this kind of gas did not represent nearly as important an economical factor as the blast-furnace gas.

This is beyond doubt partly explained by the fact that coke-oven gas was always available in a thoroughly clean condition, so that dust troubles were not encountered. After the question of economically burning coke-oven gas had been solved, the inventors of these burners turned their attention to blast-furnaces gas, cleaned and cooled blast-furnace gas being available at a large number of plants. As the difficulty of dirt accumulation in the internal-flue boiler cannot be overcome, even at the present time, it is still held in Europe that for the purpose of obtaining good boiler efficiencies it is necessary to clean the gas.

In both design and appearance, nearly all of the first German burners all followed the Bunsen type, and were mostly round in shape, partly, perhaps, because that shape was obviously the proper one for the circular flues of the internal-flue boilers. The chief disadvantage of the cylindrical burner, namely, the varying friction with the varying gas pressure, did not figure very largely, since, owing to the cleaning plants (which as a rule almost entirely absorb the already low top pressure in European plants), "boosters" to restore the pressure of the gas were necessary anyhow, and provided a fairly uniform, gas pressure. The second disadvantage of the Bunsen type of burner, namely, that the number of units varies with the size of the boiler, was of much smaller importance in Europe, since the small (usually 100 to 150 hp.) boilers did not as a rule call for more than two burners. Developments in the United States differed materially from those in Europe. In the first instance, the boilers almost universally used here are water-tube boilers of much larger size, and were all originally equipped with two burners, so that the natural tendency of the American blast-furnace operator and steam engineer was to adhere to the two burners.

The Bradshaw Burner.

A. N. Diehl¹ has described a number of types of American burners, so that I can content myself with a few words about the development of the Bradshaw burner, to which this paper, among other things, is devoted.

G. D. Bradshaw realized from the beginning the difficulty of pressure differences, which would result in differing amounts of friction, and likewise differing percentages of air aspiration. While nearly all the other burners have flat or tapering mixing tubes, he followed the Venturi principle and gave the burner the general shape of a Venturi tube. This principle is as follows:

In the flow of any fluid in a pipe or stream the sum of the static or pressure head and the dynamic or velocity head at any two sections is a

¹ A. N. Diehl: Modern Methods of Burning Blast-Furnace Gas under Stoves and Boilers, *Monthly Bulletin*, American Iron and Steel Institute, vol. iii, No. 10, p. 265, (October, 1915).

constant, if the friction and eddy-current losses between the sections be neglected. That is,

$$h_1 + \frac{v_1^2}{2g} = h_2 + \frac{v_2^2}{2g}$$

where h_1 and v_1 are the pressure and velocity at the first section and h_2 and v_2 the same at the second section. If h_1 be practically the atmospheric pressure, h_2 may be made less than the atmosphere by a proper and not prohibitive change in section between the two points. When a burner embodies this principle, the subatmospheric pressure h_2 may be employed to draw air into the stream of gas. Should the quantity of gas change, the difference between the gas pressure and the atmospheric

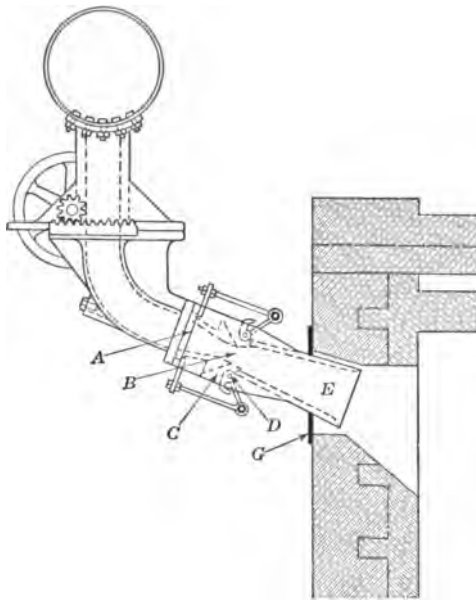


FIG. 1.—CUT THROUGH BRADSHAW BOILER BURNER.

pressure increases in proportion to the difference in the squares of the former and the new velocities. Since the flow of gas through an orifice is proportional to the square root of the pressure difference, the air drawn in by the suction varies directly with the gas flow, and, once adjusted, the ratio of gas and air is kept constant. All of this is predicated upon a reasonably constant total head, including both pressure and velocity heads. The total head at any point may be kept closely constant by preventing great changes in the pressure of the gas and the draft in the combustion chamber. The gas pressure will vary with the irregular operation of the furnace, but the short path of the gas through the burner and its small losses therein make the combustion draft by far the greater

factor in the maintenance of a constant condition within the burner itself. This draft can be held constant by any accepted type of balanced draft regulator.



FIG. 2.—STIRLING BOILER WITH BRADSHAW BURNER.

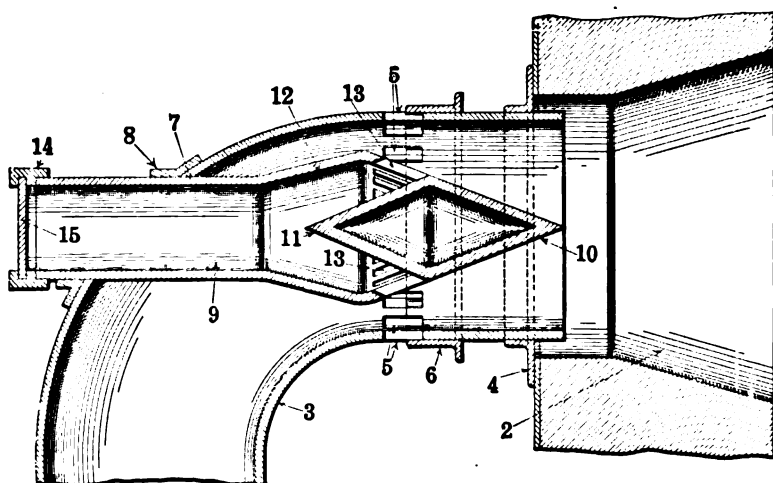


FIG. 3.—CUT THROUGH BRADSHAW STOVE BURNER.

Figs. 1 and 2 clearly show the general principles of the Bradshaw boiler burner. The gas enters at A into the converging section of the burner by which its velocity is gradually increased as it approaches the throat B. At B are placed the air-inlet openings C, which are regulated

by the adjustable scroll dampers *D*. From this point onward the burner is diverging and the velocity of the mixture of gas and air is reduced by the expanding tube *E*, which ultimately discharges into the combustion chamber. At *G*, additional secondary air enters around the tube *E*. The burner is designed in such a manner that the velocity of gas and air in the expanding tube *E* is sufficiently high to prevent back flashing. Should a large reduction in the quantity of gas cause back flashing, the return of the gas supply automatically forces the flame back out of the discharge tube.

In the stove burner, where the round shape had of necessity to be adhered to, Bradshaw secured the operation of the same principle by inserting a bullet-shaped cone in the inside of the burner tube, which also gives him the converging entrance passage of the gas and diverging outlet passage therefrom to the outlet part, as shown in Fig. 3.



FIG. 4.—BRADSHAW STOVE BURNER BULLET.

The Bradshaw stove burner is a reversal of the boiler burner in this respect: that the primary air is admitted internally through the air tube 9, as well as externally through the holes 5, shown in Fig. 3. The air regulation is obtained by slide 15 and the annular ring 6. Fig. 4 shows one of the parts of the Bradshaw stove burner. It will be noticed that in other respects both the shape and the general design of the present stove burner are adhered to.

PRINCIPLES OF COMBUSTION

The principles of combustion are of course generally known; but I insert them here as an introduction to a few useful tables concerning the composition of waste gases. The combustion of gas, as everybody knows, is the chemical reaction of the combustibles in the gas with the oxygen contained in the combustion air. Each kind of gas requires a certain definite quantity of oxygen for complete combustion. If too little air is admitted combustion must remain incomplete; as a matter of fact, it

will nearly always be impossible to obtain complete combustion for any length of time, without a certain amount of excess air, since the mixture of gas and air is rarely if ever sufficiently intimate to effect complete combustion unless an excess of oxygen is present. In any case it is not safe to burn the gas with less than from 5 to 10 per cent. excess of air, because the variation in the composition of the gas render the danger of incomplete combustion too great.

An excess of air is also necessary, because combustion without such an excess of air, particularly if all the air is primary air, has a tendency to be explosive, and creates eddies which interfere with the aspiration effect of the burner.

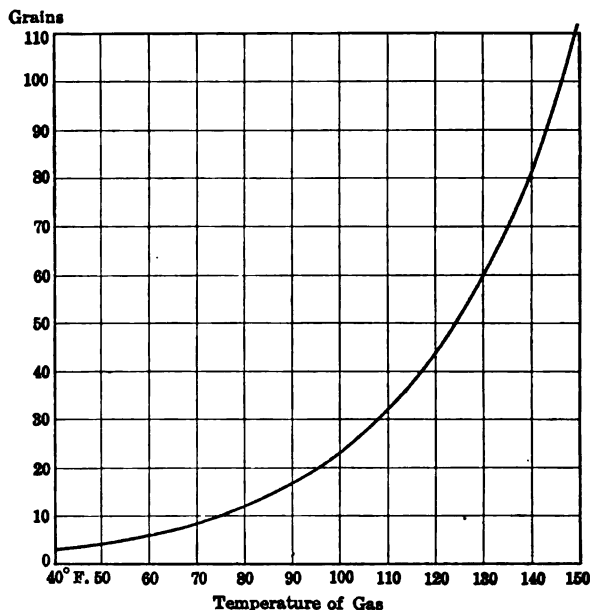


FIG. 5.—GRAINS OF VAPOR PER CUBIC FEET OF DRY GAS MEASURED AT 60°F. AND ATMOSPHERIC PRESSURE IN SATURATED GAS AT DIFFERENT TEMPERATURES.

The volume of oxygen necessary in order to burn completely a given volume of gas is found as follows:

$$\text{Total oxygen} = \frac{(\text{CO} + \text{H}_2)}{2} + 2\text{CH}_4$$

I mention these three constituents as they are the only ones ever present in blast-furnace gas. The composition of the waste gas providing complete combustion without excess air will then be:

$$\text{CO}_2 = \text{CO}_2 + \text{CO} + \text{CH}_4$$

$$\text{H}_2\text{O} = \text{H}_2\text{O} + \text{H}_2 + 2\text{CH}_4$$

$$\text{N}_2 = \text{N}_2 + \frac{\text{Total Oxygen} \times 79.5}{20.5}$$

The composition of the waste gas from blast-furnace gas containing 25 per cent. CO, 13 per cent. CO₂, 3 per cent. H₂, 0.4 per cent. CH₄, 58.6 per cent. N₂, when burnt without excess of air will then be as follows:

$$\begin{array}{r} 38.4 \text{ parts CO}_2 \\ 3.8 \text{ parts H}_2\text{O} \\ 116.0 \text{ parts N}_2 \\ \hline 158.2 \text{ parts} \end{array}$$

The theoretical CO₂ on dry waste gas will therefore be 24.9 per cent. As with complete combustion without excess of air each cubic foot of gas will yield 154.4 cu. ft. of dry waste gas of which 24.9 per cent. is CO₂, it is simple to calculate the actual volume of the waste gas from the percentage of CO₂ present.

It will be noticed that in all these calculations the furnace gas has been assumed to be dry. Since, however, fully dry blast-furnace gas never exists, it is in each instance necessary to ascertain the amount of moisture in the gas. This is done by finding the dew point of the gas, that is, the temperature at which the water commences to condense. Fig. 5 shows the amount of water vapor per cubic foot of dry gas, measured at 60°F. and atmospheric pressure, in saturated gas at different temperatures.

CAUSES OF LOW BOILER EFFICIENCY

There are four causes of low boiler efficiency which I will review in the order of their importance: (1) incomplete combustion; (2) combustion with large excess of air; (3) water vapor in the gas; and (4) deficiencies in the boiler plant itself.

The first three can be combined in one by saying "low combustion temperature." Deficiencies of the boiler plant itself (for instance air leakages in the front part of the boiler) are also frequently the cause of low combustion temperatures. Nevertheless, high combustion temperatures are quite compatible with low efficiencies due to faults of the boiler plants, such as cracks in the rear of the boiler settings, broken baffles, scale, dust accumulations on tubes, etc.

Low Combustion Temperature

It is well known that every gas has a certain highest combustion temperature which is obtained if the gas is fully dry and burned without excess of air. This temperature lies for most blast-furnace gas between 2,450 and 2,500°F. for cold gas, and between 2.8 and 11.8 per cent. higher for hot gas of a temperature of 200 to 600°F. It must be the aim of the gas engineer to get as near as possible to these theoretical combustion

temperatures. If the combustion temperatures are right, the rear temperatures will take care of themselves. It is an accepted rule that for

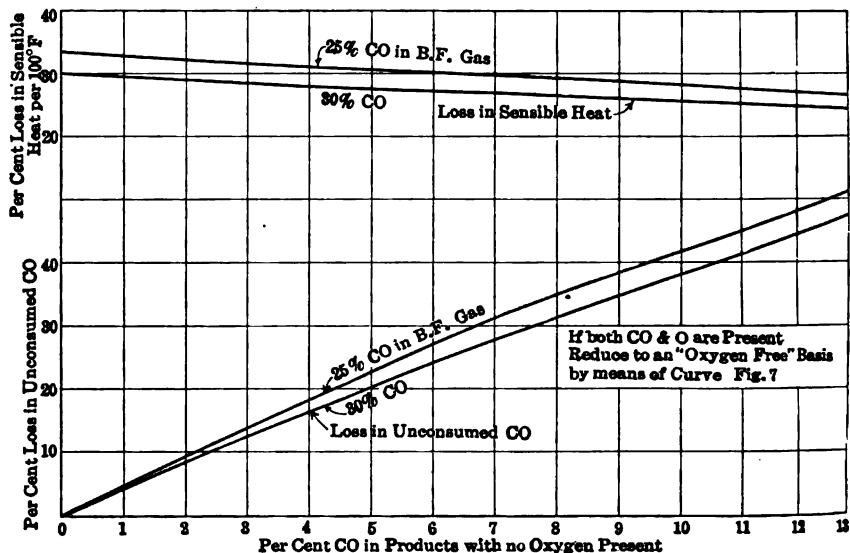


FIG. 6.—LOSSES THROUGH INCOMPLETE COMBUSTION.

any given gas (provided the same quantities of gas are passed in a given time unit) the rear temperature will always be in an inverse relation to the combustion temperature. This will show at once the importance of

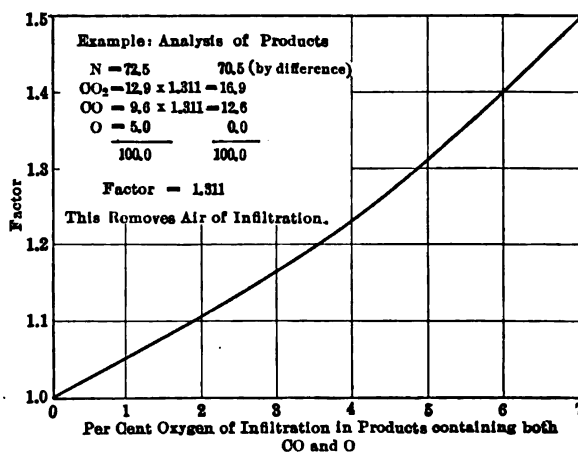


FIG. 7.—FACTORS FOR CORRECTING ANALYSES.

combustion with a low excess of air as a large excess will not only increase the volume but also the temperature of the waste gas. All this is predicated on otherwise faultless boiler conditions.

Fig. 6 shows in what manner the efficiencies are affected by unburned CO, giving the losses of both sensible and latent heat through incomplete combustion, for gases containing 25 and 30 per cent. CO with 0 to 13 per cent. CO in the waste gas, without excess of air.

If the analysis of the waste gas shows excess of oxygen in spite of the unburned CO, both CO₂ and CO must be multiplied by a certain factor which depends on the amount of oxygen in the waste gas. Fig. 7 gives these factors for amounts of oxygen varying from 0 to 7 per cent.

Example: the analysis of waste gas is	N ₂	74.8
	CO ₂	20.6
	CO	0.6
	O ₂	4.0
		100.0

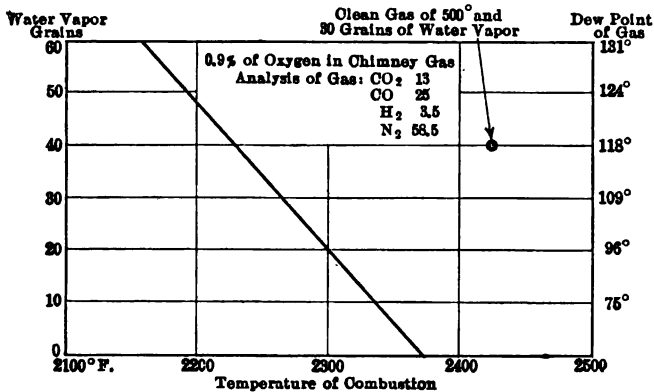


FIG. 8.—TEMPERATURES OF COMBUSTION OF BLAST-FURNACE GAS BURNED WITH AIR AT 60°F. AND 10 PER CENT. AIR SURPLUS. GAS SATURATED WITH DIFFERENT AMOUNTS OF WATER VAPOR PER CUBIC FOOT OF DRY GAS MEASURED AT 60°F. AND ATMOSPHERIC PRESSURE.

For 4 per cent. O₂, the factor is 1.23 and by multiplying CO and CO₂ by this factor we obtain the following oxygen-free analysis:

$$\begin{array}{rcl}
 \text{N}_2 & = & 73.924 \text{ (by difference)} \\
 \text{CO}_2 & 206 \times 1.23 = & 25.338 \\
 \text{CO} & 6 \times 1.23 = & 0.738 \\
 \text{O}_2 & & = 0.0
 \end{array}$$

The loss due to the unburned CO can then be ascertained from Fig. 6 as 2.8 per cent. for gas with 30 per cent. CO, and the total loss through sensible heat 2.98 per cent. for each 100°F. rise in temperature over boiler-house temperature. If stack temperature is 500°F. higher than

the boiler-house temperature the total loss is 17.7 per cent. This table is correct for all gas containing CO + CO₂, 38; H₂, 3; and N₂, 59 per cent., the dew-point being 120°F. and the gas temperature 400°F. If used for other gas it must only be considered a rough-and-ready way for ascertaining the approximate stack losses.

Fig. 8 shows how the combustion temperature is affected by water vapor in the gas. It gives the temperature of combustion of blast-furnace gas burned with air at 60°F. and 10 per cent. air surplus, and carrying different amounts of water vapor per cubic foot of dry gas measured at 60°F. and atmospheric pressure.

This curve is particularly interesting, since it answers the question, to what temperature cleaned blast-furnace gas should be cooled, in order to give the best efficiency. It will be admitted that without exceptionally cold water the gas cannot be cooled below 75°F., but even this temperature will not often be obtained, as in most instances it is preferable to use the cooling water from the blast furnace for washing purposes so that the possible temperatures will usually lie between 80 and 100°F. The difference in combustion temperature will not be greater than about 40°F. or less than 2 per cent. The heat abstracted by the water vapor in the waste gas is also trifling. Assuming the waste-gas temperature to be 600°F., then the loss as compared with the gas of 70°F. will be as follows, allowing for the gain through sensible heat of the gas at the high temperatures:

Gain in efficiency as compared with gas of 70°F.	
For 80°F. gas temperature.....	0.06 per cent.
For 90°F. gas temperature.....	0.08 per cent.
For 100°F. gas temperature.....	0.03 per cent.
Loss of efficiency as compared with gas of 70°F.	
For 110°F. gas temperature.....	0.10 per cent.
For 120°F. gas temperature.....	0.34 per cent.
For 125°F. gas temperature.....	0.56 per cent.

This shows that as long as the gas temperature is kept approximately at about 100°F. the best possible results will be obtained and that it does not pay to use enormous quantities of water in order to reduce the gas temperature below this point.

Sensible Heat.—A few remarks will here be opportune about the importance of the sensible heat of the gas. The opinion has frequently been propounded that cold, clean gas is always preferable to hot unwashed gas. This is true for stoves which cannot be cleaned while they are working and on which hot uncleaned gas cannot be used to best advantage. If here gas is burned with a small excess of air, the high combustion temperatures will bring about fusing of the dust and consequent slagging in the stoves. In the case of water-tube boilers, however, where a blowing of

the tubes about once in every turn will effectually remove the dust from the tubes the proposition is not correct.

In order to be in a position to ascertain the value of the sensible heat in the gas at a glance it will be useful to consult Fig. 9 which shows curves giving the thermal capacity of the various constituents of blast-furnace gas from 60 to 1,100°F. This curve together with some of those mentioned above has been kindly put at the author's disposal by A. Steinbart of the National Tube Co., Pittsburgh.

As an example, let us take a gas of the following analysis:
CO₂, 13; CO, 26; H₂, 3; CH₄, 0.5; and N₂, 57.5 per cent. at a temperature

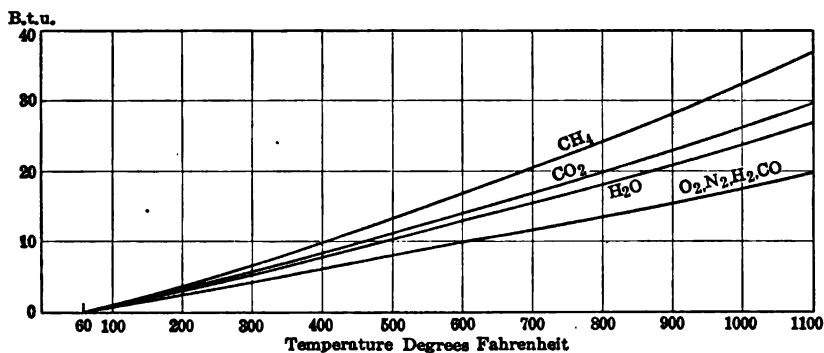


FIG. 9.—THERMAL CAPACITY OF GAS AT DIFFERENT TEMPERATURES.

of 450°F. By referring to the figure we find that each cubic foot of CO₂ contains 9.8 B.t.u., CO, H₂ and N₂, 7.2 B.t.u. and CH₄, 11.6 B.t.u. The total sensible heat is therefore as follows:

CO ₂	0.13 × 9.8	1.274
CH ₄	0.005 × 11.6	0.058
Balance	0.865 × 7.2	6.228
Total		7.560 B.t.u.

It cannot be sufficiently emphasized that a hot gas containing, say, 107 B.t.u., both in latent and sensible heat, is just as valuable as cold gas of 60°F. containing 107 B.t.u. in latent heat alone—at least as far as gas-fired water-tube boilers are concerned. The loss of sensible heat on account of washing is therefore a net loss which can in no way be recovered, even if we assume that the cost of washing the gas would be counterbalanced by the cost of cleaning the boilers. In a later part of the paper it will be shown that there is no difficulty in obtaining a boiler efficiency of 75 per cent. and more with uncleaned gas. A boiler efficiency of 75 per cent. on hot gas would correspond with an efficiency of 80.5 per cent. on cold gas and unless this could be exceeded, which I very much doubt, there could be no profit in gas washing.

Fig. 10 shows the losses through sensible heat on account of washing.

Fig. 11 shows how the combustion temperature is affected by the sensible heat when burned with an excess of air of both 10 and 50 per cent.

Results published by Mr. Diehl as regards the National Tube Burners at McKeesport are very interesting, as they show that high efficiencies are in exceptional cases compatible with low initial temperatures. I refer to the low-combustion temperature of about 210°F. They are in fact lower than they ought to be when the composition of the burned gas is considered. This composition, the same as the stack temperatures, leaves nothing to be desired. I explain these results by the fact that lingering combustion takes place and that the course of the products of combustion through the boilers is so unusually long that this lingering

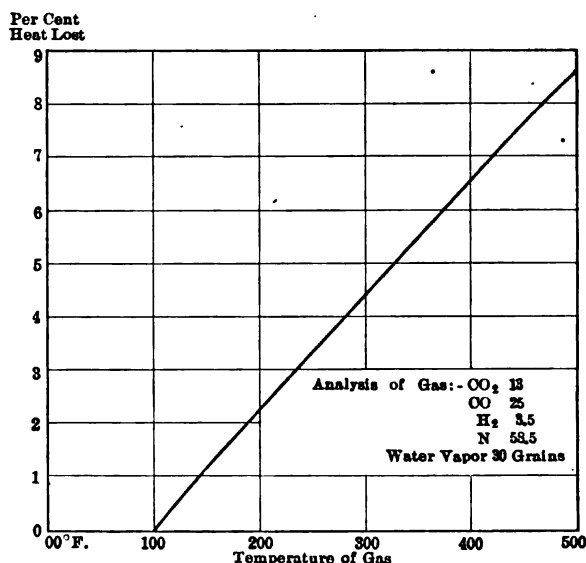


FIG. 10.—LOSSES THROUGH WASHING.

combustion which in 99 out of 100 cases is absolutely fatal to the efficiency of the boilers, does not make itself felt through a concurrence of exceptional circumstances which will rarely be repeated. In any case we have here the extraordinary fact of a burner which itself gives a comparatively low efficiency and yet results in a good boiler efficiency.

Water in Charge.—Hitherto we have only contemplated the effect of water added to the gas after it leaves the blast furnace. From the point of view of combustion alone, it is, of course, immaterial where the water is added, whether in the furnace or in the washing plant. Nevertheless, it will be interesting to note how a wet charge affects the gas economy of the furnace. Let us assume that the dew-point of a certain blast-furnace gas is 130°F. By referring to Fig. 5 we find that each

cubic foot of gas carries 60 grains of moisture. This water must not only be evaporated, but also brought to the top temperature of the furnace. Assume the latter to be 400°F. and that each ton of coke is responsible for 150,000 cu. ft. of gas measured at 60°F. and atmospheric pressure.

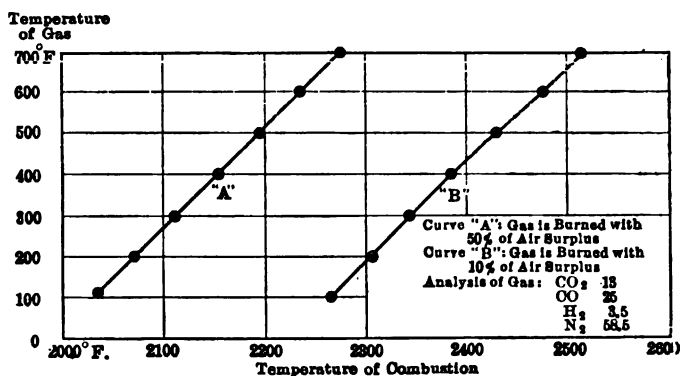


FIG. 11.—COMBUSTION TEMPERATURE WITH VARYING EXCESS OF AIR.

For each ton of coke 9,000,000 grains (1,285 lb.) of water must be evaporated and brought to a temperature of 400°F. These must first be raised to 212°F. which will consume 1,285 (212 - 60) = 195,100 B.t.u. The evaporation heat is 1,285 × 966 = 124,000 B.t.u. The 1,285 lb. of water

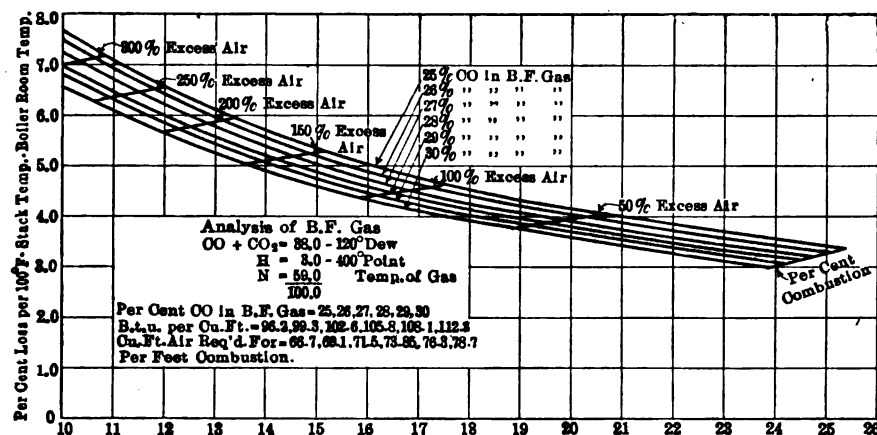


FIG. 12.—PERCENTAGE LOSSES THROUGH STACK GASES BASED ON CO₂ CONTENTS.

vapor are equal to 27,000 cu. ft. at 60°F. From Fig. 9 we find that the thermal capacity of water vapor from 212 to 400°F. is approximately 4.75 B.t.u. per cubic foot. This heat will be required to raise 1 cu. ft. to 400°F. so that 27,000 cu. ft. will require 128,500 B.t.u. in all. The total heat is thus:

	B.t.u.
To bring water to 212°F.....	195,100
To evaporate water.....	1,240,000
To heat vapor to 400°F.....	128,500
Total, per ton of coke.....	1,563,600

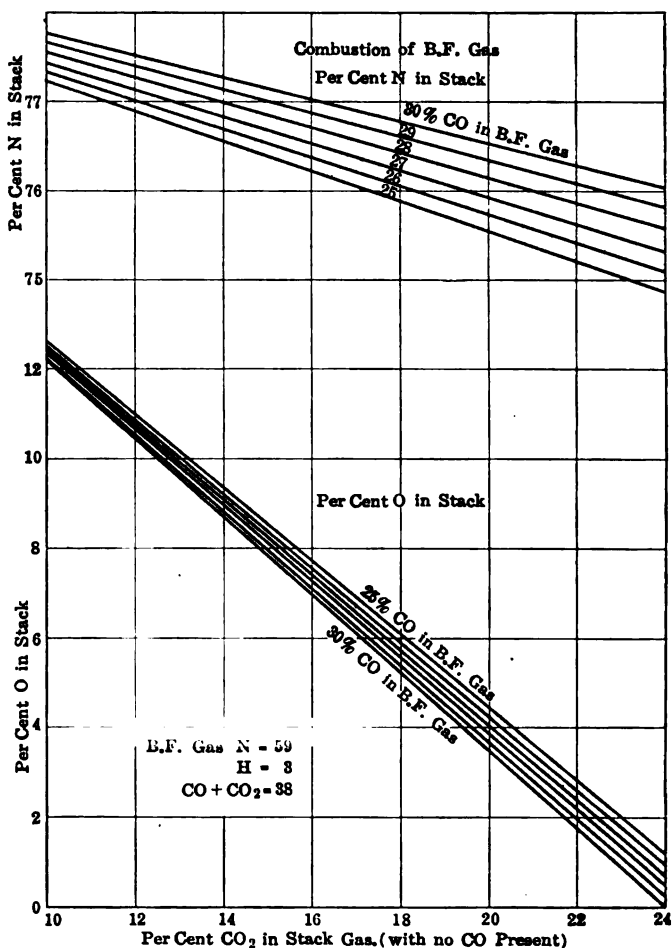


FIG. 13.—OXYGEN AND NITROGEN IN STACK GAS CORRESPONDING TO VARIOUS PERCENTAGES OF CO₂ FOR COMPLETE COMBUSTION.

If the calorific value of coke is 12,500 B.t.u., then 125 lb. of coke out of every ton used is employed in heating and evaporating the water.

While it is impossible to eliminate all this moisture, and while its total elimination would result in disturbances in the working of the furnace, owing to high top temperatures, one ought to allow the top temperature to reach the highest point which is compatible with the smooth

working of the furnace and not unnecessarily reduce it by damping the ores. For the sensible heat in the gas is just as important a factor for steam raising as the latent heat. With the various types of sintering plants there is no difficulty in recovering the valuable constituents from flue dust. The increased quantity of flue dust due to a dry charge need not deter the operator. That a fairly dry charge can be run has been proved by J. S. Fraser, at Monessen, Pa., who, although using large quantities of Messabi ores, does not damp his ores when charging the furnace.

Other Losses.—To complete the diagrams which are necessary in

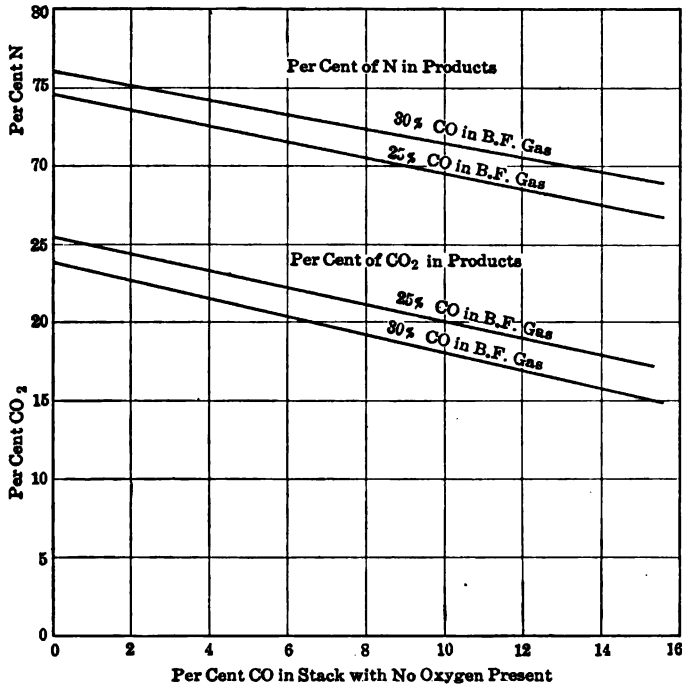


FIG. 14.—CARBON DIOXIDE AND NITROGEN IN STACK GASES CORRESPONDING TO VARIOUS PERCENTAGES OF CO₂ IN OXYGEN-FREE STACK GAS.

order to read at a glance the losses through waste gas as revealed by the stack analysis I offer Figs. 12, 13 and 14.

Fig. 12 gives the percentage losses for each 100°F. rise of temperature over the boiler-house temperature, for varying CO₂ in the waste gases from furnace gases containing from 25 to 30 per cent. CO.

Figs. 13 and 14 are useful as checks on the correctness of Orsat tests. Fig. 13 shows the percentages of oxygen and nitrogen corresponding to the various percentages of CO₂ for complete combustion; Fig. 14 the nitrogen and carbon dioxide corresponding to the various percentages of CO for oxygen-free stack gas.

Bad Boiler Conditions.—Bad boiler conditions and the losses resulting

are: defective brickwork, causing air leakages; unsubstantial brickwork, causing high radiation and conduction losses; lack of boilerhouses, with the same results; defective baffles; scale; or too short passages for products of combustion. The remedies are obvious.

The whole secret of economy is to be able to know at any given moment what one is getting and what one ought to get. To start with the latter, blast-furnace operators cannot be too urgently advised to draw up once a week, or at least every month, a carbon balance sheet of their furnaces calculating the quantity of gas from the carbon in the furnace charge. If these balance sheets are to be of any value it will be necessary to take at least one gas analysis a day and also ascertain the dew-point of the gas every day; if the gas is washed, to keep reliable records of the gas temperature as well.

Air Infiltration.—How seriously boiler efficiency is impaired by air infiltration is shown by the following experiment made by Mr. Bradshaw at Johnstown. Mr. Bradshaw admitted false air into the boiler at various distances from the burner and before starting his experiment saw to it that he had the boiler and burners in otherwise perfect working condition.

Boiler No. 132, March, 1915

CO ₂	Rear Temperature	
11.7	626°F	} Gas pressure 4.5 in.
13.0	620°F	
13.3	616°F	
18.8	578°F	
18.8	575°F	
19.7	577°F	
15.0	580°F	} Gas pressure 3.5 in.
18.0	545°F	

This experiment shows that air infiltration not only causes a loss, because all this air has to be heated to the stack temperature, but also results in an increase of the latter, or, in other words, reduces the heat absorption capacity of the boiler tubes. This is due to the fact that the thermal conductivity is in direct relation to the difference of temperature between the gases and the outside temperature of the boiler tubes, so that if the temperature of the products of combustion is reduced, the conductivity itself is reduced and less heat is transferred. This again shows how the boiler efficiency is affected by the combustion temperature; and this is also the explanation of the fact that in a good boiler 70 per cent. of the total evaporation should be done in the first third of the boiler, 20 per cent. in the second and 10 per cent. in the last third. For this reason, air infiltration is a serious factor wherever it takes place and whether or not it interferes with the initial temperature. It is erroneous

to assume, for instance, that air infiltration which takes place in the rear passage will only tend to reduce the stack temperatures correspondingly to the amount of heat absorbed by the cold air. In a case where the stack temperature was 550°F. and the CO₂ 18 per cent. with complete combustion (the low CO₂ being due to air infiltration in the rear of the boiler), the CO₂ after eliminating this air infiltration, was raised to 22 per cent. without an increase in the stack temperature. This proves that even at this point air infiltration means a net loss to the boiler.

Blowing the Tubes.—Another fact which must be constantly kept in mind is the necessity of blowing the tubes. Fig. 15 shows the stack temperature of a gas-fired boiler. The tubes were blown only once in

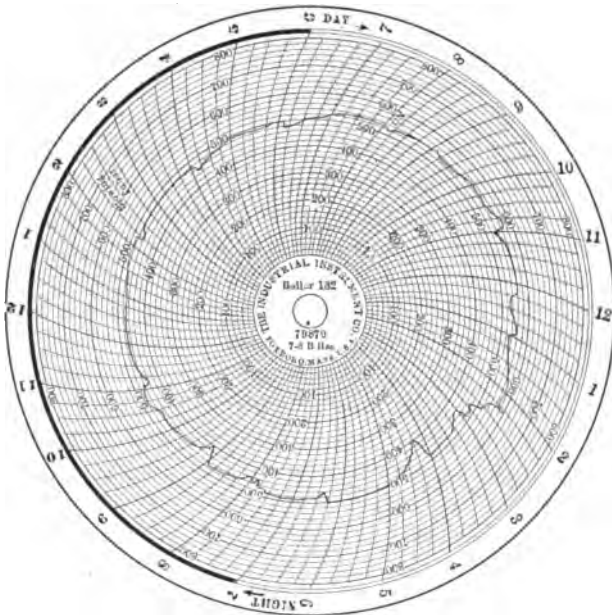


FIG. 15.—STACK TEMPERATURE CHART SHOWING EFFECT OF BLOWING TUBES.

24 hr. and it will be noticed that as a result of the blowing of the tubes the stack temperatures dropped by 150°. On account of these experiments, the tubes were blown every 12 hr. which resulted in a constant drop of temperature of 100°, a reduction by one-fifth of the loss through waste gas. An experiment made by the Pittsburgh Steel Co. illustrates how the load of the boiler is affected by the accumulation of dirt on the water tubes. When the first test was run the boiler tubes had not been blown for three days and the result was that the maximum average load obtainable from a 500-hp. boiler during 8 hr. was 720 hp. Subsequently the boiler tubes were carefully cleaned and another test of 8-hr. run and the result was that instead of 720 hp. 827 hp. was obtained,

with even a decrease of losses through stack gas. In both cases the boilers were run at the maximum load compatible with good efficiency.

Boiler-Loads.—A few words as regards the boiler loads may be of interest. We, of course, all know that the official rating of 34.5 lb. per horsepower does not at all represent the actual capacity of the boiler. In this connection it may be interesting to say something about European customs. In England, water-tube boilers of the Stirling or Babcock-Wilcox type are rated for normal working conditions at $4\frac{1}{2}$ lb. of actual steam per square foot of heating surface, which is equivalent to about 150 per cent. rating under the American rules, and the users of such boilers naturally expect to get a considerable overload out of them. In Germany, Belgium, and France, I do not think it would be possible to get an order for a water-tube boiler with a smaller guaranty than 6 lb. actual per square foot of heating surface, so that their rating really represents 200 per cent. of the American rating.

The results obtainable from internal-flue boilers are even more astonishing. I have run boilers of that kind at $8\frac{1}{2}$ lb. actual per square foot of heating surface, equivalent to 9.75 lb. from, and at, 212°F. , which would represent an overload of 282 per cent. for American practice. As a matter of fact, the flexibility of a modern boiler is very much greater than is generally assumed, and high efficiencies are quite compatible with loads of 200 per cent. and more. This load depends entirely on the chimney draft; and the limit is not reached until all the dampers are wide open and balanced draft is established in the combustion chamber, at which draft the boilers get all the gas and air which the stack can handle. A very simple and effective means to increase the boiler capacity, which has been resorted to in a good many cases, is to install induced draft. In Europe a number of modern boiler plants have been built without stacks, with fan draft only, and it has been possible to obtain astonishingly high boiler loads.

The maximum boiler load is, of course, to a certain extent dependent on the dust in the gas, and the limit is reached as soon as it is no longer possible to keep the water tubes clean by blowing once every turn. This limit will probably lie at about 180 per cent. At least the Cambria Steel Co. has been able, by blowing once every turn, to run its boiler plant for many months at a load of 165 per cent. while maintaining excellent efficiencies. This load can be exceeded without difficulty if the gas is cleaned. As a rule, the cleaning of boiler gas will only be resorted to if, owing to lack of space, new boilers cannot be installed, since it would be impossible to show any saving even if the boiler capacity should be increased by cleaning, say 20 per cent. Take the case of a 500-ton furnace. With suitable burners, a furnace of this size is capable of generating 5,100 rated boiler-hp. per hour. To generate this power at 165 per cent. load, 3,100 hp. boiler capacity are required. If the load were raised by gas cleaning

to 200 per cent., 2,600 rated boiler-hp. would do the work, so that a 500-hp. boiler would be saved. But the gas cleaning would result immediately in a drop in the evaporation of about 7 per cent. or 357 boiler-hp. At \$25 per boiler-horsepower per year, the actual loss through cleaning would be approximately \$8,900 a year which alone would be sufficient to cover the expenditure for an additional 500-hp. boiler quite apart from the cost of a cleaning plant which would require at least another \$20,000.

Method of Measuring Economy.—There has been considerable discussion about the means of ascertaining whether or not one is getting good economical results. By some of the operators it is maintained that the only reliable method is to ascertain the boiler efficiency by stack analysis and temperature, and assume a certain percentage for radiation and conduction losses; other operators, and especially a number of steam engineers, hold that this method is all right for occasional tests, but does not answer in cases where large numbers of boilers are operated, and it is used to keep track of the actual results obtained from month to month and from year to year. These men, whose opinion I favor, hold that a monthly figure giving pounds of steam per pound of coke is the one and only reliable method of tracing results. The chief objection to this method of ascertaining the efficiency of a boiler plant is, that a large percentage of the gases are used for the stoves, and in a good many cases some of the gases are also used in gas engines. I hold that occasional efficiency tests are very misleading indeed. It is not very difficult to obtain good efficiencies for a given period of a few hours or days, covered by a test. The main difficulty is to maintain such efficiencies over an indefinite period. I make bold to say that with any kind of a burner, as long as it is possible to admit sufficient air for combustion, we can obtain excellent efficiencies for a period during which a boiler is under constant supervision and is continuously regulated, but we shall first find, that as soon as the supervision is relaxed (as it must be, once the test is finished) the efficiency will fall back forthwith; and second, that the ordinary type of burner will have a comparatively low maximum load, and the stack losses will be much larger with the ordinary type burner at a high load than with a modern type burner, such as the Bradshaw.

The contention that pounds of steam per pounds of coke does not give any reliable information on account of the gas used by the stoves can easily be met. In the first place, as the stoves get the first supply of gas and the boiler only the surplus, there is a very simple method of seeing to it that the stoves always get the same quantity of gas. All that has to be done is to put a pressure gage in the boiler house and instruct the boiler foreman to keep a uniform gas pressure at the stoves, which will bring about a uniform gas consumption. Better still take up stoves and boilers together and equip them both with suitable combustion arrangements, giving the highest possible efficiency. If that has been

done, the pounds of steam per pound of coke during any one month give the joint results of both stoves and boiler; and if they fall below that amount which corresponds to the coke ratage, then it must be found out by efficiency tests whether boiler or stoves are responsible for the reduction in the steam output.

If any part of the gas should go to gas engines, a measuring device should be installed and, in ascertaining the results which ought to be obtained, due allowance for the gas consumption of the gas engine should be made. As a rule it will even suffice to take the total output of electricity for the month and allow 18,000 B.t.u. per kilowatt—or whatever the consumption may be in the case.

I reproduce here a table published by S. M. Marshall, late Chief Engineer of the Cambria Steel Co., showing the evaporations to be obtained from blast-furnace gas at the various rates of pounds of coke per ton of iron:

Coke per Gross Ton of Iron, Pounds	Cold Gas, Steam per Pound of Coke, Pounds	Hot Gas, Steam per Pound of Coke, Pounds
1,736	3.56	3.85
1,800	3.61	3.90
2,000	3.73	4.03
2,200	3.86	4.17
2,240	3.89	4.20
2,400	3.99	4.31
2,600	4.12	4.45
2,800	4.24	4.58
3,000	4.37	4.72

The steam is given in pounds from and at 212°F. Mr. Marshall figures on a very low stove efficiency of only 57 per cent. If this were raised to 75 per cent. the above figures would be increased by 8.95 per cent.

These figures conclusively answer in the negative the question, whether it pays to wash blast-furnace gas for boilers. Take the case of a 500-ton furnace using 2,000 lb. of coke per ton of iron. The loss through cleaning is $\frac{1}{5}$ ton of steam per ton of coke. If the average value of a ton of steam is 25c., cleaning for boilers results in a loss of 5c. for every ton of coke used, quite apart from the cost of cleaning, which is only to a very small extent offset by the saving in wages for cleaning boilers.

Efficiency Tests.—If the above evaporations are not reached, efficiency tests on both stoves and boilers should be made. The losses sustained are made up in both cases of stack losses, and radiation and conduction losses. The stack losses may be in sensible heat alone if complete

combustion takes place, or in both latent and sensible heat if the combustion is imperfect.

Stack Losses.—The stack losses must be ascertained through stack-gas analysis and temperature measurements. The stack gases should show, under total absence of unburnt CO, from 22 to 23 per cent. CO₂ and the temperature, even for high boiler loads, should not exceed 550°F. These results, of course, are obtainable on good modern boilers only, although even with imperfect boilers, like Cook, Cahall and Wheeler, efficiencies of 70 per cent. and more, when running on gas, are obtainable in everyday practice.

With the assistance of Figs. 6, 7, 12, 13, and 14, the stack losses of both stoves and boilers can be easily ascertained. It should not be overlooked, however, that particularly with bad combustion arrangements, the CO₂ is a much varying quantity, so that conclusions should be drawn only from a large number of analyses. It will, therefore, always be advisable to use a CO₂ recorder and take Orsat tests for unburned CO at least every 5 min. for several hours.

Where mixed coal and gas firing is used, a corresponding allowance for the water evaporated by the coal should be made. If there is any doubt as to the amount of water evaporated by the coal, special tests for the purpose of ascertaining the evaporation factor to be used can easily be arranged.

In this connection it may be pointed out that the practice of mixed firing is anything but desirable. Where the boilers are equipped with combustion chambers of sufficient size, and a good combustion arrangement is available, pilot firing, even in the case of the one-furnace plant, is not necessary. The Cambria Steel Co., for instance, has run 20 B & W boilers, equipped with Bradshaw burners and good-sized Dutch ovens, without pilot firing of any kind, in spite of the fact that these boilers received their gas from one furnace only. The heat stored in the combustion chamber was high enough to re-ignite the gas, even if it had stayed away for 10 min. or more, during a cast or change of tuyère. In this case, however, the furnace boilers were connected up to the mill boilers so that there was no danger of losing the steam pressure. At many of the one-furnace plants, this connection does not exist, and pilot firing cannot be dispensed with, since at any moment it may happen that the gas gives way and the power has to be supplied by coal exclusively. In that case, it should be kept in mind that the boilers are primarily gas-fired boilers and that all that is necessary is to maintain a small fire on the grate. It is essential, that after coal has been thrown on, the fire and ash-pit doors should be sealed up again, to avoid infiltration of air, and that where the boilers are equipped with blowers, these should only be used while actual coal firing is necessary, and sealed up at all other times. If for any reason the gas is not sufficient to carry

the load, it is advisable to spread the gas over a certain number of boilers which are not coal-fired except in emergency cases, and do the coal firing on other, separate boilers. In view of the very different conditions under which coal and gas have to burn, the advisability of this procedure is self-evident.

Boiler Construction.—We may inquire next, how boiler efficiency is influenced by the boiler construction as apart from the condition of the setting. It has generally been held that the one-pass boiler, such as the Cahall, Cook or Wheeler, is absolutely hopeless from the point of view of efficiency, and when I first approached the question of improving the efficiency of this type of boiler I did so with no little misgiving. I do not claim that I am in a position to say definitely to what extent a one-pass boiler can be worked economically; but, in experiments made at the Upson Nut Co.'s plant in Cleveland, I was surprised to find that boilers of this type could be run on a 100 per cent. load with very high efficiency. In some cases this was as high as 75 per cent., but all attempts to overload this kind of boiler with any show of efficiency proved futile, in spite of ample draft.

The chief disadvantages of the one-pass boilers are the great difficulty of keeping them air-tight and their very large radiation and conduction losses in comparison with their capacity. In any case, it is interesting to know that it is possible to run a boiler of this kind on gas at rated load with a good efficiency. Where, therefore, ample boiler capacity in one-pass boilers is available, it would certainly not pay to install modern boilers for the purpose of increasing efficiencies.

As regards boilers of the modern type, there can be very little doubt that, from the point of view of efficiency, it does not matter so much which type is used. Equally good efficiencies have been and are obtained on all types of boilers, provided the combustion arrangements are up-to-date.

Boiler House.—A very important feature in the operation of the gas-fired boiler, as we have seen already, is the "moral" of the boiler house. It would be a grave mistake to assume that the adoption of a good combustion arrangement alone would solve the difficulty. That will go a long way toward better efficiencies; but wherever efficiencies in excess of 65 per cent. are desired, the supervision of brickwork settings and the regulation of the draft are of paramount importance. We have already considered the great influence of air infiltration.

Draft.—The second factor which is of great importance is the draft. That a burner of the type which employs a preliminary mixture of gas and air provides its own combustion air, independently of the chimney draft, is only partially true.

The aspiration velocity of any burner is chiefly dependent on the chimney draft. Mr. Buessel, of the American Steel & Wire Co. of

Cleveland, has made some interesting experiments on an installation of Bradshaw burners at the Upson Nut Co.'s plant. He inserted a pipe, flattened at the end, into the air port of the Bradshaw burner and measured the aspiration velocity in this pipe by means of an anemometer. It is evident that owing to the friction in this pipe the figures obtained do not give positive information but they are invaluable as giving the relative velocities for various boiler drafts. With the same amount of gas in both cases, Mr. Buessel found that for a chimney draft of 0.35 in. the aspiration velocity in the air ports was 888 ft. per minute and that after reducing the chimney draft to 0.17 in. it fell to 713 ft. per minute. It is evident that a burner which receives sufficient air for a given quantity of gas at one aspiration speed must have suffered from incomplete combustion for the same quantity of gas at lower speeds. The aspiration velocity stands in a direct relation to the velocity in the mixing chamber, provided that all friction which tends to absorb the aspiration effect is eliminated, as in the case of the Bradshaw burner. In order to keep the relation between gas and aspirated air constant, the pull which is effected on the velocity of the gas and air mixture by the chimney draft must be kept constant, or the quantitative relation between gas and air will be upset. The importance of draft regulation will of course depend on the variation in gas pressure, so that, for instance, at a place where these variations are small, draft regulations will not play nearly as important a part as in places where there are frequent and large differences in gas pressure. Draft regulation is particularly important where it is desired to burn the gas with a very low excess of air; for, with dampers in a stationary condition, an increase in the gas pressure must of necessity result in incomplete combustion. I have found that it is not safe to raise the CO_2 higher than 21 per cent. without automatic draft regulation whereas 24 per cent. can safely be maintained with such regulation. From gas containing 25 per cent. CO the loss through waste gas with 24 per cent. CO_2 at a stack temperature of 560°F . is 17.75 per cent. while with 21 per cent. CO_2 it is 20 per cent. (see Fig. 12). This difference of 2.25 per cent. may not appear important, but it must be kept in mind that with a lower initial temperature the stack temperature in the case of 21 per cent. CO_2 will also, as likely as not, be higher.

Supervision.—Then there is the item of supervision. If the dampers are automatically governed, as for instance by the McLean governing device, and Bradshaw burners are used, then the sole duty of the boiler tender will be to watch the settings. He will not have to watch either burners or dampers. Without automatic damper governing, he will have either to do hand regulation or regulate the air valves for the purpose of maintaining an adequate quantity of air. Such regulation

of the air-valves, particularly in large plants would be practically out of the question.

As regards air-tight boiler settings, it may be useful to know that a very satisfactory coating is obtained by means of whitewash and cement. After several coatings have been given to the boiler, the settings will get a smooth surface. The falling off of the coating at any part will at once be detected and can be repaired as it occurs.

In addition to watching the draft, even with the most perfect combustion arrangement it is necessary to check the composition of the waste gas and ascertain the temperatures from time to time. One or two analyses per week for the boilers will be quite sufficient and, at least with the Bradshaw burner, resetting of the air ports will as a rule only be necessary where a change in the iron is made, with a consequent considerable change in the composition of the gas.

RESULTS WITH THE BRADSHAW BURNER

In conclusion, I wish to say a few words about the results obtained with the Bradshaw burner. I give below a list of 52 stack analysis taken at the Cambria steel works, which show an average of 21.4 per cent. CO_2 under total absence of CO , a stack temperature of 581°F . and a boiler load of 165 per cent. As the gas in question contains 25 per cent. CO and the rise in temperature over the boiler house temperature is about 500° , the total losses through waste gas are 20 per cent., so that the boiler efficiency, even if the losses through radiation and conduction were as much as 5 per cent., would be 75 per cent.

I might mention that with boilers in a boiler house, it is extremely unlikely that the radiation and conduction losses would be as high as 5 per cent. As far as I am aware, no reliable figures of these losses are available at the Cambria Steel Co. I have personally conducted tests on 500-hp. boilers in England and found that the radiation and conduction losses of these boilers, which were in a boiler house, were less than 2 per cent. The German periodical, *Feuerungstechnik*, published in 1914 a table of experiments made on 12 Stirling boilers, working in various parts of the United States, which showed highest losses of 2.5 and lowest losses of 0.75 per cent. through radiation and conduction.

It may be of interest to know that the dampers were hand-regulated at the Cambria steel plant, which accounts for the comparatively low CO_2 . What can be done with automatic damper regulation is exhibited by the two charts, Figs. 16 and 17, one of which shows the gas pressure, the other the CO_2 in the waste gas, as given by a CO_2 recorder. The correctness of this recorder was constantly checked, and unburned CO was not found in any of the samples.

With regard to Figs. 16 and 17 attention should be given to the uni-

Cambria Steel Co. Analyses and Stack Temperatures of Waste Gases

Boiler No.	CO ₂	O	CO	N	Stack Temperature	
2-18	20.0	4.5	0.0	75.5	635	} Tubes not blown for 24 hr.
2-18	21.0	3.4	0.0	75.6	650	
2-19	21.7	4.0	0.0	74.2	615	
11-126	23.2	2.0	0.0	74.8	445	} February, 1915, load 145 per cent. of rating.
11-127	22.5	2.3	0.0	75.2	440	
11-138	21.2	3.2	0.0	75.6	525	
11-137	22.8	1.6	0.0	75.6	525	
11-136	21.6	2.6	0.0	75.6	515	
11-135	20.4	4.0	0.0	75.6	555	
11-129	21.0	3.5	0.0	75.5	515	
11-130	22.0	3.0	0.0	75.0	485	
11-120	21.5	2.3	0.0	76.2	580	
11-121	20.0	3.8	0.0	76.2	570	
11-138	21.4	2.6	0.0	76.0	515	
120	22.6	1.4	0.0	70.0	576	
121	20.7	1.2	1.1	76.8	619	
122	23.3	0.4	0.0	76.3	568	
123	22.6	0.4	0.0	77.0	499	
124	21.0	3.0	0.0	75.7	569	
125	20.8	4.2	0.0	75.0	537	
126	19.5	4.9	0.0	75.6	627	
127	21.6	1.8	0.0	76.6	580	
129	19.4	3.8	0.0	76.6	568	
130	21.4	0.6	0.2	77.8	610	} May, 1915, load 165 per cent. of rating tubes blown 12 hr. before.
131	22.8	2.2	0.0	74.9	518	
132	22.2	1.8	0.0	76.0	565	
133	21.0	2.0	0.0	77.0	655	} Continued May, 1915, load 165 per cent. rating tubes blown 12 hr. before.
134	20.2	1.0	0.0	78.7	610	
135	21.6	2.6	0.0	75.8	585	
136	22.6	0.8	0.0	76.6	542	
137	22.8	0.6	0.0	76.6	655	
138	19.8	3.6	0.0	76.6	585	
139	21.0	4.6	0.0	74.4	595	
Average of last	19	21.4	2.08	0.0	76.3	581
120	23.2	0.8	0.0	76.0	557	} June, 1915, tubes blown from 6-12 hr. before taking temperatures 165 per cent. rating.
120	22.0	1.3	0.0	76.0	560	
123	22.4	0.6	0.0	77.0	510	
120	20.8	1.8	0.0	77.4	555	
120	21.0	1.2	0.0	77.8	560	
131	20.4	3.0	0.0	76.6	565	
131	21.4	3.2	0.0	75.4	585	
131	20.8	1.4	0.0	77.8	570	
124	20.6	3.4	0.0	76.0	565	
124	20.8	3.2	0.0	76.0	580	
131	22.6	2.8	0.0	74.6	520	
131	22.6	1.6	0.0	75.8	540	
131	22.8	0.8	0.0	76.4	550	
131	22.3	1.0	0.0	76.7	548	
121	20.4	3.2	0.0	76.4	530	
94	26.0	0.4	0.0	73.6	530	
94	26.2	0.2	0.0	73.6	535	
122	23.2	0.6	0.0	76.2		
129	20.2	4.8	0.0	75.0		

form CO₂, which runs from 22 to 23 per cent. in spite of the greatly varying gas pressure, *e.g.*, between 6 and 8 a.m. and 10 and 12 p.m. The rear temperature was only 495°F. This is, however, explained by

the fact that the boilers were only running on rated load. The gas in question contained 25 per cent. CO and by referring to Fig. 12 we find that the loss through waste gas was 3.6 per cent. for every 100°F. (stack temperature — boiler room temperature) or a total of 15 per cent., so that the total boiler efficiency, assuming radiation and conduction losses at 5 per cent., is about 80 per cent.

An interesting test on a large load was run by the Pittsburgh Steel Co. and A. N. Diehl jointly at Monessen. I give below the complete

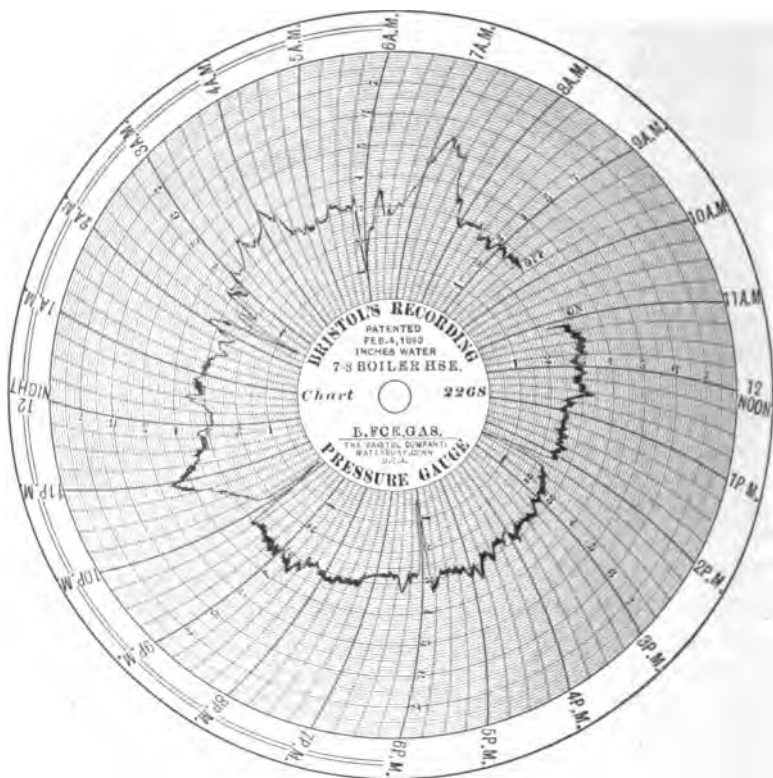


FIG. 16.—GAS-PRESSURE CHART.

results of this test as they were put at my disposal. It must be kept in mind that this test was for the purpose of ascertaining the highest load compatible with good boiler efficiency, as will be seen from the low draft in combustion chamber of only 0.003 in.

The boiler used was equipped with a modification of the Bradshaw burner designed by J. S. Fraser. As I have already explained, it is rather dangerous to run a boiler on its maximum load, since one is constantly verging on incomplete combustion, as was the case in the present instance. Mr. Fraser's modification which is, as will be seen from Fig.

18 the application of the stove-burner principle to the boiler burner, gave nevertheless excellent results. By comparing the analysis in the combustion chamber with the stack analysis it will be seen that there was considerable air infiltration at the back of the boiler, which might of course have been stopped, but for certain reasons this was not done. The total evaporation during 8 hr. was 230,000 lb. water from and at 212°F. which is equivalent to a boiler load of 166 per cent.

The calorific value was stated as 102.8 B.t.u. of cold gas measured at

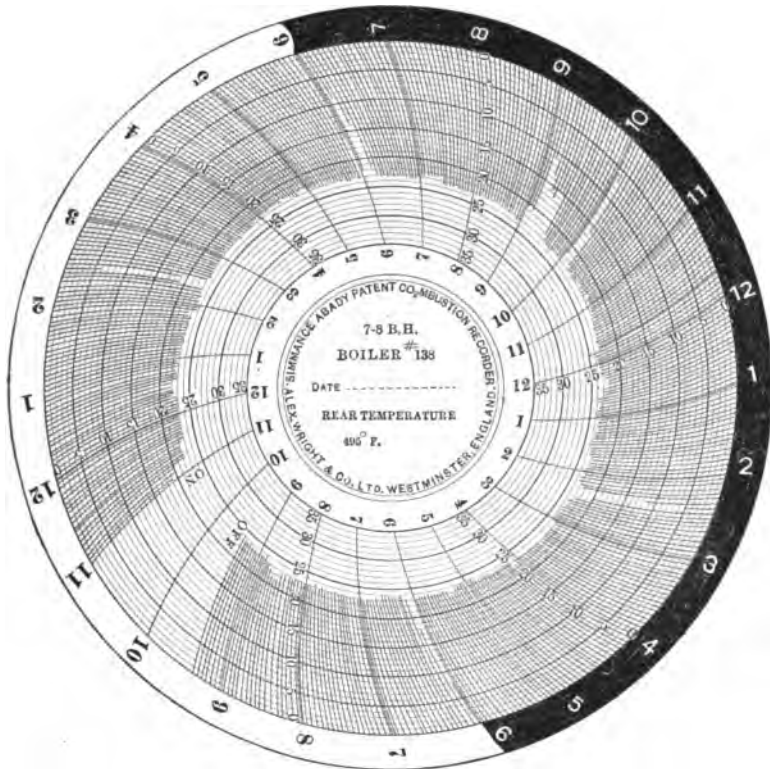


FIG. 17.—CARBON DIOXIDE CHART.

32°F. This is 97.6 B.t.u. measured at 60°F. for cold gas and 105.2 B.t.u. for gas of 377°F. measured at 60°F.

The average stack losses, in accordance with the readings obtained by Mr. Rhodes of the Carnegie Steel Co. have been, through unburned CO, 1.46 B.t.u.; through sensible heat, 20.32 B.t.u.; total, 21.78 B.t.u. equal to 20.7 per cent.

Assuming the radiation and conduction losses at 5 per cent., the total efficiency is 74.3 per cent., whereas in accordance with Mr. Merwin's figures this efficiency has been 72.3 per cent.

It is interesting to note the astonishingly high combustion temperatures which show that the burner efficiency falls little short of 100 per cent.

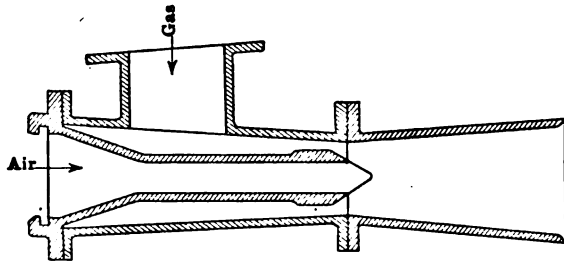


FIG. 18.—LONGITUDINAL SECTION OF FRASER'S MODIFICATION OF BRADSHAW BURNER.



FIG. 19.—STIRLING BOILER WITH FRASER'S MODIFICATION OF BRADSHAW BURNER.

The importance of the Bradshaw burner as a coal-saving device was shown very clearly by the results obtained at the plants of the Pitts-

Tests Made at Monessen

22

Gas Analysis							Rhodes Gas Analysis							Merwin Stack Analysis																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																		
Time	CO ₂	CO	H ₂	CH ₄	N ₂	B.t.u.	Combustion Chamber				Stack			Gas, Temp.	Gal. of Water	Time	Temp.	CO	CO ₂	O ₂	Draft																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																											
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burgh Steel Co. and the Upson Nut Co. during the tests run to prove guarantees. At the Pittsburgh Steel Co.'s plant, the evaporation in the best months previous to the installation of our burners had been 3.6 lb. per pound of coke. For the purpose of proving guarantees a test of four days was run, which showed a maximum evaporation on one day of 4.42 lb. and an average evaporation of 4.25 lb. per pound of coke.

At the Upson Nut Co., previous to the installation of the Bradshaw burner, the average coal consumption over a year had been 1145 tons per month. This coal consumption was reduced to 100 tons per month, a quantity which is actually required for carrying their boilers over the casts. At the same time the Upson Nut Co.'s is now constantly bleeding considerable quantities of gas.

CONCLUSION

Boiler efficiencies depend on both efficient burners and good boiler "morals." If either of the two is lacking, high efficiency will generally be impossible. Boiler "morals" include proper conditions in the boiler plant itself, as well as intelligent working and close supervision. The regulation of a boiler plant should be done, as far as possible, exclusively on the dampers. With otherwise perfect conditions, efficiencies depend on the boiler loads. With a boiler load of 100 per cent. an efficiency of 80 per cent. is possible. With higher loads, it will soon drop to 75 per cent., but as a rule there will be no need to decrease the efficiency any further. Gas washing is profitable for stoves, but almost always results in losses where it is done for boilers. Moisture in the gas does not seriously affect efficiencies, except in the case of entrained moisture, which should be avoided. Where gas washing is resorted to, the most favorable temperature from the viewpoint of efficiency is 100°F. It is advisable regularly to draw up carbon balance sheets and keep a check of the results obtained by reducing them to pounds of steam per pound of coke.

I beg to acknowledge the assistance kindly given to me, in the preparation of this paper, by O. S. Fraser and Grant D. Bradshaw, and also by Alfred Steinbart, as author of many of the curves.

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Pennsylvania Fire Clay

BY L. C. MORGANROTH, PITTSBURGH, PA.

(New York Meeting, February, 1916)

CARBONIFEROUS CLAYS

FROM a geological standpoint, but scant attention has been paid to fire-clay beds. Only within the last few years have they been the subject of individual investigation, prior to this time having been considered only in their relation to coal seams. While the relative importance of coal and fire clay is still immeasurably in favor of coal, yet an industry as large as that of fire clay is entitled to a measure of attention.

A discussion of the Carboniferous clays is practically a discussion of the clay beds in the Appalachian basin, since more than 90 per cent. of these clays mined are from this region. The clays are confined to the northern end of the basin, or the States of Pennsylvania, Ohio, Kentucky, Maryland and West Virginia, ranking in importance about in the order named.

The clay deposits of the Carboniferous period have been divided into two classes; those found in the Pottsville conglomerate series of rocks, and all those that are found in the formations above, for the reason that the clay beds above the Pottsville series all resemble each other in properties, composition and appearance, and all are distinctly different from the bed found in the Pottsville formation. The bed of clay found in the latter formation is unique in that flint clay is practically restricted to it. (A variety of flint clay is occasionally found in these upper deposits but this is in such small amounts as to be negligible.)

Carboniferous clays occur between stratified rocks; sandstones, limestones, etc., in the same manner as coal veins. They are often accompanied by a coal vein although where the clay is of a workable thickness, the coal is generally too thin to work. The clay beds are sometimes known by the name of the coal seam at which horizon they occur. Clay beds may be from a few inches up to 20 ft. in thickness. They may consist of all soft clay, all flint or both soft and flint.

The Carboniferous clay beds of Pennsylvania are the most important of any of the clay beds found in the country, and are among the oldest ones worked. We know more about this clay geologically for the reason

that Pennsylvania coal veins are better known and, as previously stated, the clay beds accompany coal veins.

Pottsville Conglomerate Series

In the State of Pennsylvania there are 16,000 square miles containing the coal-measure formation. It is within this area that the Conglomerate clay is found.

Only a comparatively small area of the Pottsville Conglomerate series lies above water level. While clay of good quality is comparatively scarce, it is still sufficiently plentiful to make it unprofitable, to mine the clay below water level because its great irregularity would necessitate a large and unwarranted development expense.

The accessibility of a clay field is another point. While large bodies of clay of the proper quality will warrant a considerable expenditure in railroads to reach it, there are limits; the average required to supply any ordinary sized brick yard is not large, and unless other freight could be handled on the same road, it is doubtful if it would pay when the length of track to be built is over 10 or 15 miles. With the above points in mind, it can be understood why the various clay mines are grouped within comparatively narrow bounds.

It is not to be supposed that the entire area within the Pottsville Conglomerate is underlaid with clay. No doubt clay beds suffered as great a loss from erosion by streams and glaciers as did the coal veins, so that today we have but a small percentage of the clay originally deposited. Again, the clay beds never covered as great an area as the coal veins. Their sedimentary origin would preclude this idea. While we look for clay within the boundaries of a certain formation, it is not to be supposed that it will be found anywhere within this boundary; the major portion of the area probably never contained any.

We know of no clay being mined in the Pottsville series of the anthracite field. It would not pay to work it by itself, if found, for it would nearly always be below water level. In the counties in the north and northwestern part of the State where the Pottsville Conglomerate outcrops, there are localities where the clay shows, but as a rule, the clay is not good. Other counties, it can be seen, will not pay even to investigate on account of lack of railroad facilities.

The bulk of the Conglomerate fire-clay mines are in Clearfield, Clinton, and Cambria Counties, with a fewer number in Somerset, Bedford, Clarion, and Armstrong. The most valuable bodies of this clay are those found in central Pennsylvania, viz.: Clearfield, Cambria, and Clinton Counties. These counties rank in the order given, in reference to quality and quantity of the clays produced. In a general way, the clays of central Pennsylvania are about the same over the entire area;

they look alike, analyze about the same, and are similar in formation of the deposit, while the brick made from them are about the same in refractoriness; yet in this area there are clay bodies that possess distinctive features and must be treated differently in their use. Chemically, the difference may be slight, if any, yet must be taken into account by the manufacturer. Clearfield County particularly possesses several bodies of this Conglomerate bed which show marked differences.

It may be that some of the Conglomerate beds now worked occur at different geological horizons, but it is impossible to determine at present if such is the case. All the Pottsville Conglomerate beds now worked in Pennsylvania are considered as occupying the same geological position, and only when the fields have been more thoroughly prospected will this fact be determined positively.

The exact position of the bed is supposed to be directly beneath the Upper Mercer Coal.

Probably the most important clay field in Pennsylvania, if not in the United States, is that at Woodland, Clearfield County, Pa. These mines are among the oldest, if not the oldest, in the State. The brick made from them are probably more widely known than any other brand. The Woodland clay is noted for its purity and for its regularity in quality. Other mines may have as good clay at times, but no mine can show for the same number of years the same regularity in quality. It has also been one of the most regular in regard to quantity; while it has been worked for a number of years, the operators claim there is more clay in sight today than there was 10 years ago.

One of the principal reasons for the superiority of the Woodland clay is the excellent quality of its soft clay. Other Conglomerate beds may have good flint, but it is the exception for them to have any amount of good soft clay. The Woodland vein averages 2 ft. of both flint and soft clay. The clay outcrops for but a short distance, probably not more than 2 miles in length. A stream running east and west cuts across the field so that there are two crop lines. Its exposure at Woodland is due to a north and south anticline which has lifted the measures above water level. The dip of the anticlinal east and west is rapid, so that in a short distanced the clay horizon is 100 ft. or more below water level.

Two other important clay fields are found in proximity to the Woodland bed; these fields are known locally as the Morgan Run field and the Andrews Creek field. The former lies about 12 miles south of Woodland and the later about 10 miles west of it. A circle of 2 miles radius will more than embrace either field. Geologically these veins occur at the same horizon as the Woodland clay but the clays differ from that of the Woodland bed. The clay of the Morgan Run field is rougher in appearance and looks more like burley than flint. The bed is thicker, but there is very little of soft clay, so little that the vein is often mined "run-of-

mine" and no effort made to keep the two varieties separate, as in the Woodland bed. The bed when used run-of-mine seems to have a right amount of bonding property. The appearance of the bed, and the fact that it is so used, indicates that the clay is not as hard as that of Woodland flint and because of this, the small amount of soft material found in the bed is sufficient to make the bond.

Anderson Creek Field.—The clay of the Anderson Creek field is distinctive inasmuch as the bed has all the appearance of a soft clay, or rather a "hard soft." It does not resemble the burley clay of the Morgan Run field nor the Woodland soft or hard, but appears to be in a class by itself. While it has not the flint-like nature of the hard and while closely resembling soft clay, yet it is as deficient in bonding properties as the best flint. Chemically, it is as pure as flint and in tests almost as refractory. A portion of this same field, locally known as the Widemire field, is a distinct variety of clay. A considerable amount of nodule clay is found in this field.

There are several other clay fields in Clearfield County which are known by local names, but they possess no distinctive characteristics. They resemble the clays of the other fields that have been discussed; in fact, the fields described resemble each other very closely; in any of them can be found clay similar to the others. The differences herein mentioned are noticeable only by close scrutiny. In a comparison with clays of other sections, the clays of Clearfield County would be taken as a whole.

Cambria County Field.—The clay mines in this county are restricted to a few localities, Dean and Blandburg being the principal mining centers. The deposit at Dean is almost all flint. There is, on an average, about 6 in. of "soft" below it, but it is often impure and is not used. The vein at Dean appears to be the flintiest and hardest of all the clays of Pennsylvania, if flint clay is a hardened form of fire clay, as previously argued; at this mine it has reached its highest development, as practically all the vein is flint clay.

The vein at Blandburg so closely resembles Woodland that little more need be said about it. There is a locality in this section where the vein consists of from 7 to 10 ft. of nearly all "soft." This soft is practically the same as the Woodland soft.

Clinton County Clays.—There are a number of separate localities where clay is mined in this county, but the character of the deposit is practically the same and one description will apply to all. The bed consists principally of soft clay; the flint clay averages from 12 to 18 in. thick. The soft clay is generally 4 to 6 ft. in thickness, and in some places even thicker, but it is not all good. It rapidly increases in impurities as the bottom of the bed is reached. The flint clay is probably as good as any, but it is the soft clay that makes these clays rank below

the best. Generally, about 1 ft. of "soft" immediately below the flint is very good, but if only the flint and this 1 ft. of soft clay were mined, the clay would be too expensive to use. As it is, a portion of the inferior "soft" is also mined to give a vein of workable thickness. Of course, there are exceptions to this statement, and mines are found where excellent soft clay is mined, but as a general rule the soft is impure. The impurity is iron.

Clarion Field.—This field takes its name from the county in which it is found. The field produces a very pure flint clay, but the clay of this grade occurs irregularly in pockets. A considerable portion of it is used in glass pot construction, some of it being calcined at the mines for this purpose. The clay is lighter in color than other Pennsylvania flint clays. Another peculiarity is the large amount of iron balls found in the vein; they range in size from that of a walnut to a football, and many are perfectly round. The amount of clay shipped from this field is dwindling, due to exhaustion.

Climax Clay.—In the vicinity of Climax and St. Charles on Red Bank Creek, Armstrong County, flint clay is mined. While this field is close to the Clarion fields, the vein more resembles the Savage Mountain clays. The vein is from 6 to 12 ft. thick, with from 2 to 2½ ft. of flint. It is more pockety than Clearfield County clays. The flint occurs more in the form of lenses in the soft clay, in a similar way to the upper-bed flints. While the soft clay is generally below, it may be on top, or both on top and bottom of the flint. Often the flint is missing entirely, and the bed is all soft.

Savage Mountain Clay.—This is the name applied to the bed of clay occurring in the Pottsville series in the southwestern part of the State. The clay field is a canoe-shaped one with its northern extremity near Williams Station, Bedford County and extending in a southwest direction into Maryland and West Virginia. The bed is distinctly of basin form dipping toward the center axis east and west as much as 25° in places. At some of the mines in this district, long tunnels are driven to meet the bed. This is possible as the outcrops occur high up on a mountain. By coming down the side of the mountain several hundred red feet and driving a tunnel, the clay is cut at a lower level. It is exceptional in clay mining to go to this expense, as the irregularity of clay deposits does not warrant it. The axis of the basin also dips toward the south. Geologically the clay is supposed to occupy the same horizon (Upper Mercer Coal) as the flint clay beds already mentioned, but the clay differs in appearance from the clay mined in central Pennsylvania. Physically it has a banded or foliated appearance which makes it resemble the so-called upper beds of flint clay that are occasionally found in central Pennsylvania. Chemically it runs higher in silica than central Pennsylvania flints, and is less refractory.

Upper Measures

Pennsylvania.—The plastic clays that accompany the coal veins above the No. 12 series are probably nearly all worked, more or less—if not for firebrick, for sewer pipe, building brick, stoneware, etc.

One of the most used of these clays is that occurring beneath the Clarion coal. It is mined extensively on the Allegheny River near the town of Kittanning, the clay in this locality being known as the Allegheny or Kittanning clay, and in Clearfield County near the towns of Bigler and Wallacetown, where it is known by the name Bigler and Wallacetown clay. The bed is from 6 to 12 ft. thick—all soft clay. Here and there a small lens of flint clay is found, but so infrequent and the lens so small as to be of no importance. The upper and middle portions of the bed are most pure; the lower often contains an excess of silica and considerable iron. In the manufacture of firebrick this clay is used mostly to serve as a bond to hold the more refractory flint clay together. The Allegheny plastic clay is used extensively in the manufacture of building brick—it makes a particularly white brick.

Beaver County.—The clay bed accompanying the lower Kittanning coal is by far the most important. It is the bed mined along the Ohio River in Pennsylvania, Ohio and West Virginia. The bed may be from 6 to 12 ft. in thickness; the upper portion ranging from 2 to 6 ft. is the purest and is that used in the manufacture of firebrick. Firebrick made exclusively of this fire clay would rank as a No. 2 or No. 3 firebrick. The lower portion of the bed when mined is used in making sewer pipe, stoneware, building and paving brick.

In this region, as well as in Ohio and West Virginia, clays occur at the horizons of the Brookville, Clarion, Middle Kittanning, Upper and Lower Freeport coals, but they are not worked on account of the prominence of the Lower Kittanning vein.

Gorman Clay.—Flint clay occurs occasionally at the horizon of the Gorman coal—between the Upper and Middle Kittanning coals—in Clearfield County. This is generally the bed referred to, in that locality, when speaking of the upper bed flint. The bed is entirely too irregular and uncertain as to quality and quantity to make it a profitable commercial undertaking. It may show a workable thickness on the outcrop, but rapidly thins down to nothing or changes into a soft clay.

Upper Freeport Clay.—Often known as the Bolivar fire clay from the town of Bolivar where it has been worked for a number of years. It is also mined at Salina, Pa. A characteristic of the bed is the large amount of iron it contains. Often the iron is in the form of concretions or balls, in which form they can be readily separated from the clay because of their color and weight. At other times the iron shows up as a stain on the clay, or is so finely divided as to make large portions of the bed

worthless. Some flint clay is found but it varies greatly as to thickness. The flint occurs in lenses and may be at any part of the bed, top, bottom or middle, or be absent entirely. As is usual with the upper bed flints, its analysis shows an excess of silica.

Pittsburgh Coal—Fire Clay.—At the top of the workable section of the Pittsburgh coal occurs a bed of clay 6 to 12 in. in thickness, that is used in places in making a low-grade refractory material. The bed of clay is too thin to work by itself. It is only mined in conjunction with the coal. While thin, the bed extends over a wide area. There are several manufacturing plants along the Monongahela River making use of it. The clay is known locally as horseback.

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Vacuum-Fused Iron with Special Reference to Effect of Silicon

BY T. D. YENSEN,* B. S., M. S., URBANA, ILL.

(New York Meeting, February, 1916)

I. INTRODUCTION

It is safe to say that of all the different materials that go to make up electrical machinery, iron is the most important. Upon its magnetic and electrical quality depends not only the efficiency of the machine but, to a very large extent, the cost. Aside from the mechanical strength, there are three characteristics of iron that are of importance in this connection, viz., magnetic hysteresis, electrical resistance, and magnetic permeability. In the homopolar machine, where the magnetic flux is unidirectional, permeability is the only important factor, but in all other machines the flux is reversed in one or more parts and causes losses due to hysteresis and eddy currents. The eddy-current loss depends largely upon the electrical resistance of the iron; the higher the resistance the lower the loss, under otherwise identical conditions. These hysteresis and eddy-current losses cause heating of the machine, and as they depend upon the magnetic density in the iron, this density must be regulated so that the heating shall not be excessive. If the hysteresis and eddy-current losses and the permeability of the magnetic circuit will permit an increase in the magnetic density, or, in other words, a decrease in the amount of iron, this in turn will mean a decrease in the amount of copper needed, with a consequent decrease in the copper loss. It is thus seen that, indirectly as well as directly, the above characteristics have an important bearing on both the efficiency and the cost of electrical machinery. In general, it may be said that iron for electrical machinery should have the following characteristics: (1) low hysteresis loss, (2) high electrical resistance, and (3) high permeability. These are the characteristics that investigators have been striving to attain since electrical machinery first came into use.

Until the beginning of the present century, Swedish charcoal iron was regarded as the best grade of iron for magnetic purposes, and even now it ranks high among the various grades used. About that time Sir Robert Hadfield¹ produced a number of iron alloys that revolution-

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¹ Barrett, Brown and Hadfield: *Scientific Proceedings of the Royal Dublin Society*, vol. vii, Sec. 2, Part 4, p. 67 (January, 1900). *Journal of the Institution of Electrical Engineers*, vol. xxxi, p. 674 (1901-02).

ized the iron industry as far as magnetic iron was concerned. His 2.5 per cent. silicon alloy and 2.25 per cent. aluminum alloy showed a higher permeability and gave a lower hysteresis loss than the purest Swedish iron. Perhaps of even more importance was the high electrical resistance of these alloys, reducing the eddy-current loss to one-third, or less, of that of the Swedish iron under identical conditions.

Since the time of the appearance of Hadfield's alloys comparatively little improvement has been made in the magnetic quality of iron and iron alloys. What has been done has been mostly in the way of modifications of Hadfield's silicon steel. In 1906 Prof. C. F. Burgess, of Wisconsin, commenced a series of investigations upon the magnetic and allied properties of electrolytic iron and its alloys with other elements. While the improvements made by Burgess were not remarkable, his investigations, I think, pointed out the direction in which improvements might be expected. Burgess and his associates melted their pure iron in a carbon resistance furnace in an atmosphere of carbon monoxide under ordinary pressure. It was found that by this method of melting the iron absorbed about 0.1 per cent. carbon, evidently by decomposition of the CO gases. According to Barrett² this carbon content should raise the specific electrical resistance of the iron from 10.1 to 10.6 microhms. However, the resistance for Burgess' purest iron was found to be 12.5 microhms, showing that the iron had absorbed some other impurity besides carbon. There is little doubt but that this other impurity is oxygen, because if carbon is absorbed by the decomposition of CO, the oxygen liberated would naturally combine with the iron.

The experiments on pure iron recorded by Burgess were used as the starting point of the investigation commenced by the author four years ago. Doubly refined electrolytic iron, deposited according to the methods used by Burgess, has formed the basis of the investigation. It was deposited from Swedish charcoal iron. The raw material and the product had the following composition:

	Swedish Iron, Per Cent.		Doubly Refined Iron, Per Cent.
C.....	0.1630	C.....	0.006 to 0.01
Si.....	0.0320	Si.....	0.01
S.....	0.0002	S.....	trace
P.....	none	Fe (by difference).....	99.98
Mn.....	none		
Fe (by difference).....	99.8000		

The iron was placed in crucibles made from electrically fused magnesia, and these crucibles were in turn placed in graphite crucibles and securely covered. At first a Hoskins resistance furnace was used for melting of the iron, but this was abandoned after a large number of attempts had

² *Proceedings of the Royal Society of London*, vol. lxi, p. 480 (1902).

been made to keep the iron from being contaminated. If the crucibles were left exposed in the furnace the iron was so badly oxidized that it cracked to pieces under the hammer in the attempt to forge it. If buried in crushed carbon the iron absorbed from 0.05 to 0.15 per cent. carbon, confirming the results of Burgess. A photomicrograph of the iron thus produced is shown in Fig. 1. In order to overcome the difficulties above referred to, a vacuum furnace was constructed, modeled after the Arsem type of furnace. The inside parts consist entirely of graphite, cut from solid graphite electrodes, with the exception of the water-cooled copper tubes that serve as supports as well as leads. A Geryck pump is capable of maintaining a pressure of 0.5 mm. of mercury or less with 500 or 600 grams of molten iron in the furnace. The state of the charge may be observed through a mica window in the top.



FIG. 1.—ELECTROLYTIC IRON MELTED IN AN ATMOSPHERE OF CO GAS UNDER ATMOSPHERIC PRESSURE (COMPARE WITH FIG. 11). MAGNIFICATION 40 DIAMETERS. ETCHED WITH PICRIC ACID.



FIG. 2.—TOP SURFACE OF AN INGOT OF PURE IRON, MELTED IN VACUO.

This method of melting took care of the contamination (see Fig. 11), and the iron, after being permitted to cool in the furnace, was perfectly bright when removed (Fig. 2). In magnetic quality, too, it rose far above anything known at that time.² The maximum permeability for this pure vacuum iron was found to be 19,000 as compared with 6,000 to 8,000 for the best commercial iron used at the present time. The hysteresis loss, too, was found to be much lower than for ordinary iron. An iron with very promising characteristics was thus produced. If the electrical resistance of this iron, which in the pure state is very low, could be increased without impairing its magnetic properties, an ideal iron for electromagnetic purposes would be produced.

In experimenting further with the vacuum iron, alloying it with small

² See Bulletin No. 72, Engineering Experiment Station, Illinois University (1914).

quantities of carbon⁴ in one case, and boron⁵ in another, the fact was revealed that, in spite of very careful chemical cleaning of the electrolytic iron previous to melting, the iron contained about 0.4 per cent. of oxygen in the form of some oxide of iron. This was shown by the disappearance of the equivalent amount of these alloying elements. In the case of carbon, this element would not combine with the iron unless added in quantities above 0.3 per cent. Assuming that the carbon was oxidized to CO, this amount of carbon corresponds to 0.4 per cent. of oxygen. Thus carbon would appear to be the ideal reducing agent for iron oxide *in vacuo*. The danger is, of course, that too much carbon may be added, leaving a small amount to combine with the iron, with disastrous effect upon the magnetic quality. Boron differs from carbon in its behavior upon iron, in that only part of it is oxidized by the iron oxide, while the remainder combines with the iron. The effect of the combined boron upon the magnetic properties of the iron—and upon some of the mechanical properties as well—is quite similar to that of carbon. Both elements should, therefore, be thoroughly eliminated from iron intended for high-permeability, low-hysteresis purposes. The results of these experiments suggested that if an alloying element could be found that had a strong reducing power at the same time as it had no lowering effects upon the magnetic quality, the magnetic properties of the pure vacuum iron might be improved even above what had already been attained. Silicon suggested itself as the most promising element for this purpose. The results given in this paper will show that these expectations have been fully realized. Certain percentages of silicon added to pure iron not only raise the electrical resistance to a desirable degree, but also raise the magnetic properties to a point that was not thought possible a year ago. While it is the magnetic properties that make these iron-silicon alloys particularly important, their mechanical properties, too, present characteristics that are of great interest, and these will be discussed in detail; a number of photomicrographs are also reproduced.

II. MATERIAL, APPARATUS AND METHODS⁶

1. General

As already stated, the iron used as the basis of the investigation consisted of electrolytically refined iron, containing less than 0.01 per cent. carbon and about 0.01 per cent. silicon. Before being used, the iron was crushed, cleaned with HCl, distilled water and alcohol, and then dried

⁴ *Bulletin No. 72, Engineering Experiment Station, Illinois University*, pp. 40 to 42 (1914).

⁵ *Bulletin No. 77, Engineering Experiment Station, Illinois University* (1915).

⁶ Details with regard to material, apparatus and methods used are given in *Bulletins No. 72 and No. 83, Engineering Experiment Station, Illinois University*.

by means of ether *in vacuo*. About 600 grams of this cleaned iron was then placed in a fused magnesia crucible together with the desired portion of the alloying element, covered with a magnesia cover and placed in the vacuum furnace where it was melted under a finishing pressure of 0.5 mm. of mercury. The ingots, after being allowed to cool in the furnace, were heated in an ordinary coke forge and forged into rods about $\frac{1}{2}$ in. (1.25 cm.) by 20 in. (50 cm.) under a steam hammer. No contamination took place in this operation. From these rods the test pieces were prepared; one rod 14 in. (35.5 cm.) long for the magnetic and electrical tests; two test pieces, $2\frac{1}{2}$ in. (6.3 cm.) long, with threaded ends, for mechanical tests; and one small specimen for the metallographic investigation. Magnetic and electrical tests were made and photomicrographs were taken after the following heat treatments:

(a) As forged.

(b) Annealed at 900°C. Cooled at a rate of 30° per hour.

(c) Annealed at 1,100°C. Cooled at a rate of 30° per hour.

Mechanical tests were made as forged and after annealing at 970°C. A few specimens were annealed at 1,100°C. The annealing was done *in vacuo* so as to preclude any possible contamination by gases, the vacuum part of the furnace used for this purpose consisting of an "electroquartz" tube with mercury-sealed ends.

2. The Magnetic Testing

The Burrows compensated double bar and yoke method was used for the magnetic testing, as this is the method now generally adopted whenever accurate testing is desired. Briefly stated, the apparatus consists of one main coil, and one auxiliary coil, separately controlled, and four compensating coils, connected in series. By means of three secondary coils the magnetic flux can be investigated at different points of the magnetic circuit and equalized by adjusting the currents in the magnetizing coils. With no leakage of flux, the magnetizing force at the middle of the main coil is

$$H_T = 0.4 \pi N_T I_T$$

where N_T = number of turns per centimeter of main coil

I_T = current in main coil in amperes.

An error is introduced here on account of the effects of the ends of the various coils on the magnetizing force at the center of the main coil. With the silicon rods of highest permeability the maximum compensating current was 30 times the current in the main coil, causing a theoretical error in H_T of + 2.3 per cent.⁷ However, experimental evidence seems

⁷ See *Bulletin of the U. S. Bureau of Standards*, vol. vi, No. I, Reprint No. 117.

to show that the actual errors are larger than the theoretical considerations would lead to, and may in the extreme cases amount to eight times the theoretical error.⁸

Another source of error in the magnetic testing is mechanical strain caused by clamping the rods in the yokes, but this can be avoided by careful clamping.⁸

The apparatus was calibrated from time to time by means of an air coil, and also by means of rods submitted to the Bureau of Standards for standardization. The results obtained at the University check very well with those obtained by the Bureau and also with results obtained by the calibration laboratories of two of the large manufacturing companies.

3. *Photomicrographs*

These were obtained by polishing in the usual way, using jewelers' rouge for the final polishing. Great care had to be exercised with the higher silicon alloys. Numerous small spots would appear on the surface of some of the specimens, too numerous to be caused by graphite, as the carbon content amounted to only 0.01 to 0.02 per cent. By careful polishing the spots were in most cases removed, as seen by the photomicrographs.

The etching was generally done by means of saturated picric acid in alcohol. In a few cases, however, a 10 per cent. solution of nitric acid in alcohol was found to give more satisfactory results.

The highest silicon alloys, containing 4.92 and 6.57 per cent., respectively, had to be washed with dilute hydrofluoric acid to remove the thin film of a silicon composition that invariably formed on the surface after etching.⁹

4. *Mechanical Testing*

The mechanical testing consisted in obtaining the yield point, ultimate strength, the elongation before the specimen had commenced to "neck," the ultimate elongation, and the reduction of area at the point of fracture. The testing was done on an Olsen 10,000-lb. testing machine, in which the test pieces were secured between screw sockets. The load was applied uniformly by means of an electric motor, and in most cases the yield point was detected with certainty. The test pieces from the 6.57 per cent. alloy could not be threaded on account of their brittleness and had to be tested between flat grips.

⁸ See *Bulletin No. 83, Engineering Experiment Station, Illinois University*, 1915.

⁹ L. Guillet: *Revue de Metallurgie*, p. 46, 1904.

III. RESULTS

1. *Chemical Properties*

Table I gives a complete list of all the alloys made, together with the chemical analysis for silicon. From the amount of silicon added, the percentage lost in terms of the weight of the ingot has been calculated and listed in the fourth column. The only elements that are present

TABLE I.—*List of Alloys*

Specimen No.	Silicon Added, Per Cent.	Silicon Content as per Chem. Anal., Per Cent.	Silicon Lost, Per Cent.	Remarks
3-51	0.000	About 0.001		These rods were discarded after annealing at 1,100°C. on account of oxidation due to a leak in the furnace.
3-52	0.000	About 0.001		
3-53	0.000	About 0.001		
3-54	0.000	About 0.001		
3-55	0.000	About 0.001		
3 Si 05	0.092	0.068	0.024	The magnetic data for these rods, after annealing at 900°C. were taken before the effect of strain due to tight clamping of the yokes was noticed. It was impossible to retest these rods annealed at 900° because they were already annealed at 1,100°. Data for the 1,100° annealing were taken without strain.
3 Si 06	0.185	0.148	0.037	
3 Si 07	0.276	0.242	0.034	
3 Si 08	0.368	0.309	0.059	
3 Si 09	0.460	0.400	0.060	
3 Si 10	0.551	0.472	0.079	
3 Si 11	0.643	0.563	0.080	
3 Si 12	0.734	0.673	0.061	
3 Si 13	0.825	0.698	0.127	
3 Si 14	0.916	0.822	0.094	
3 Si 15	0.138	0.064	0.074	
3 Si 16	0.046	0.010	0.036	
3 Si 17	0.230	0.230	0.000	
3 Si 18	1.810	1.741	0.069	
3 Si 19	2.690	2.550	0.140	Not forgeable. Crushed into mass of crystals.
3 Si 20	3.560	3.580	-0.020	Flaw in center of rod after forging.
3 Si 21	0.092	0.048	0.044	Not forgeable. Same as 3 Si 19.
3 Si 22	0.185	0.091	0.094	
3 Si 23	0.276	0.205	0.011	
3 Si 24	2.690	2.570	0.120	Contaminated in melting.
3 Si 25	3.560	3.400	0.160	
3 Si 26	2.260	2.180	0.080	Not forgeable.
3 Si 27	3.125	2.730	0.395	
3 Si 28	4.410	4.440	-0.030	Not used. Made from electrolytic iron that proved to be impure.
3 Si 29	5.230	4.920	0.310	
3 Si 30	8.420	8.560	-0.130	Used for mechanical tests only.
3 Si 31	2.260	1.710	0.550	
3 Si 32	6.850	6.570	0.280	Used for rings for comparative magnetic testing.
3 Si 33	0.459	0.420	0.039	
3 Si 34	0.915	0.700	0.215	Used for mechanical tests only.
3 Si 35	0.276	0.193	0.083	
3 Si 36	3.980	3.550	0.430	Used for mechanical tests only.
3 Si 37	4.810	4.390	0.420	
3 Si 38	2.460	2.260	0.200	Used for mechanical tests only.
3 Si 39	3.560	3.185	0.375	
3 Si 40	3.560	2.960	0.600	Used for mechanical tests only.
3 Si 41	2.690	2.360	0.330	
3 Si 42	2.690	2.700	-0.010	Used for mechanical tests only.
3 Si 43	2.770	2.550	0.220	
3 Si 46	2.600	2.530	0.070	Used for mechanical tests only.
3 Si 47	2.510	2.410	0.100	
3 Si 51	2.660	2.600	0.060	

in the alloys as impurities in a measurable quantity are carbon, amounting to about 0.01 per cent., and oxygen.

It has been previously shown that the electrolytic iron, even after the thorough cleaning to which it was subjected before melting, contained about 0.4 per cent. oxygen in the form of some oxide of iron. It was shown that while carbon will reduce the iron oxide before commencing to combine with the iron, boron will combine with the iron before all the iron oxide is reduced, its affinity for oxygen being about twice its affinity for iron. From Table I it is seen that silicon in this respect acts like boron, with this difference, that its affinity for iron is much stronger than its affinity for oxygen. The percentage of silicon lost, that is, the percentage that has been oxidized and changed into slag, increases, somewhat irregularly, with the silicon added, but reaches a maximum of about 0.5 per cent. As silicon oxidizes at SiO_2 , the maximum amount of oxygen absorbed is 0.44 per cent., or the same amount that was found in the two previous investigations, using carbon or boron.

2. Mechanical Properties

The results of the mechanical tests are shown in Tables II and III and graphically in Figs. 3 and 4. A comparison between these results

TABLE II.—Results of Mechanical Tests as Forged

Number of Specimen	Silicon Content, Per Cent.	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation, Per Cent.		Reduction of Area, Per Cent.	Remarks
				Before "Necking"	Ultimate		
3-39	0.001	35,800	44,700		39	80.4	Failed near punch mark.
3-53	0.001	44,400	46,500	3	24	53.8	
3-55	0.001	38,000	40,800	8	40	88.5	
3 Si 16	0.01	41,800	45,200	11	35	78.0	
3 Si 21	0.048	42,850	46,900	5	25	91.6	
3 Si 05	0.068	36,800	43,800	10	37	92.0	
3 Si 22	0.091	35,600	43,750	12	36	91.7	
3 Si 06	0.148	38,600	45,000	11	42	94.8	
3 Si 23	0.205	42,500	49,700	10	39	93.4	
3 Si 17	0.230	41,300	47,500	10	45	89.7	
3 Si 11	0.563	40,750	51,000	12	41	92.6	
3 Si 12	0.673		58,000	9	30	91.4	
3 Si 14	0.822	45,200	55,800	11	36	93.1	
3 Si 31	1.71	68,100	76,300	6	29	87.2	
3 Si 38	2.25	66,550	77,750	12	37	82.0	
3 Si 41	2.36	77,200	86,450	8	24	66.0	Slight flaw.
3 Si 47	2.41	63,500	75,800	9	23	65.0	
3 Si 46	2.54	60,300	73,400	14	16	20.0	
3 Si 51	2.63	68,700	78,500	13	24	43.0	
3 Si 43	2.65	59,000	74,700	14	34	72.0	
3 Si 42	2.78	68,250	78,000	6	21	74.0	
3 Si 25	3.40	74,500	86,300	10	31	74.7	
3 Si 36	3.55	83,400	99,300	12	23	41.3	
3 Si 37	4.39	94,000	105,000	6	6	7.5	
3 Si 29	4.92		50,250	Nil.	Nil.	Nil.	Failed at base of head. Tested between grips without being machined.
3 Si 32	6.57	5,120	5,120	Nil.	Nil.	Nil.	

and those obtained by some of the previous investigators¹⁰ is given in Figs. 5 and 6. Fig. 7 is a photograph of some of the test pieces after

TABLE III.—*Results of Mechanical Tests Annealed at 970°C.*

Number of Specimen	Silicon Content, Per Cent.	Yield Point, Lb. per Sq. In.	Ultimate Strength, Lb. per Sq. In.	Elongation, Per Cent.		Reduction of Area, Per Cent.	Remarks
				Before "Necking"	Ultimate		
3-39	0.001	16,400	36,100		61	80.9	
3 Si 16	0.010	16,050	34,900	25.0	53	81.5	
3 Si 21	0.048	20,100	35,000	23.0	48	89.3	
3 Si 15	0.064	14,750	34,100	27.0	29	45.1	Failed near head flaw.
3 Si 05	0.068	20,400	34,900	28.0	64	94.8	
3 Si 22	0.091	14,290	35,400	26.0	64	91.8	
3 Si 06	0.148	15,890	35,200	29.0	48	67.0	
3 Si 23	0.205	25,075	38,650	19.0	50	89.4	
3 Si 19	0.230	14,910	35,500	30.0	60	84.7	Broke near head flaw.
3 Si 07	0.242	18,100	38,400	25.0	60	91.3	
3 Si 08	0.309	21,650	40,400	25.0	55	90.0	
3 Si 09	0.400	26,000	42,000	20.0	55	91.0	
3 Si 10	0.472	17,340	42,750	26.0	54	91.4	
3 Si 11	0.563	25,700	41,200	0.08			Was not stressed to failure. Flaw.
3 Si 12	0.673	26,550	45,230	21.0	45	88.2	
3 Si 13	0.698	23,100	43,000	25.0	57	89.0	
3 Si 14	0.822	26,200	45,150	28.0	50	91.6	
3 Si 31	1.71	35,600	54,250	25.0	50	90.6	
3 Si 18	1.741	45,750	55,000	14.0		84.7	Failed at punch mark.
3 Si 38	2.28	42,200	63,500	23.0	50	85.0	
3 Si 43	2.36	46,200	63,600	17.0	29	51.0	
3 Si 41	2.38	43,000	64,200	26.0	50	84.5	
3 Si 47	2.38	47,500	68,700	22.5	45	78.5	
3 Si 46	2.56	47,250	64,200	14.0	18	31.5	
3 Si 51	2.59	42,500	68,200	22.0	42	82.5	
3 Si 42	2.70	48,500	61,800	6.0	6	11.3	
3 Si 27	2.73	49,600	67,800	18.0	19	15.5	
3 Si 25	3.40	57,100	77,400	15.0	21	28.7	
3 Si 37	4.39	85,000	85,000	Nil.	Nil.	1.2	Annealed at 1,030°C. in nitrogen.
3 Si 28	4.44	72,900	91,600	14.0	24	25.1	
3 Si 29	4.92	47,700	47,700	Nil.	Nil.	Nil.	Head failed.
3 Si 32	6.57	13,000	13,000	Nil.	Nil.	Nil.	Tested between grips.
Annealed at 1,100°C.							
3 Si 41	2.35	39,400	58,300	8.0	10	10.0	
3 Si 47	2.43	41,700	63,100	24.0	45	67.0	
3 Si 46	2.48	45,200	66,300	21.0	47	75.0	
3 Si 43	2.53	43,300	62,500	20.0	40	71.0	
3 Si 42	2.61	38,900	59,500	17.0	32	61.0	Slight flaw.
3 Si 51	2.63	43,300	45,600	1.0	2	Nil.	

being tested, showing very clearly the variation in the elongation and reduction of area caused by silicon.

From the data thus presented it may be stated in general that silicon

¹⁰ Hadfield: *Journal of the Iron and Steel Institute*, vol. xxxvi, p. 222 (1889).

Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903).

Guillet: *Révue de Métallurgie*, Memoirs, vol. i, p. 46 (1904).

Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

increases the strength of iron in almost direct proportion to the amount added, until the maximum strength is reached with a silicon content of about 4.5 per cent. From this point on, the elastic limit coincides with the ultimate strength, and both decrease very rapidly. The limit of forgeability lies between 7 and 8 per cent. The values for the forged condition are considerably higher than for the annealed condition, the difference varying between 10,000 and 20,000 lb. per square inch (7 to 14 kg. per square millimeter). For the 4.5 per cent. alloy "as forged"

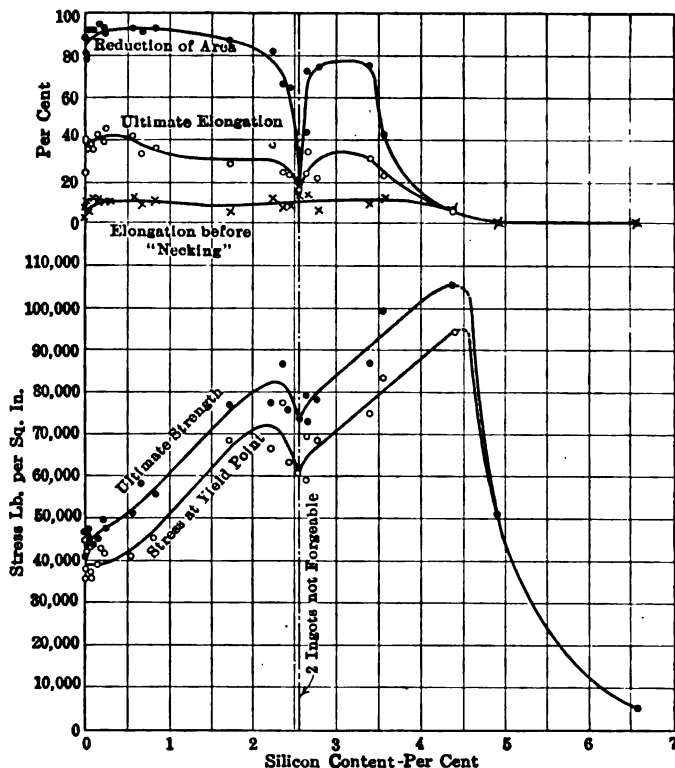


FIG. 3.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS. MELTED IN VACUO AS FORGED.

the ultimate strength is 105,000 lb. per square inch (73.5 kg. per square millimeter), about 8,000 lb. (5.6 kg.) higher than the maximum obtained by previous investigators. The practical absence of carbon in the vacuum iron causes the low-silicon alloys to be weaker than the corresponding alloys tested by previous investigators, but this same absence of carbon is evidently a cause for added strength in the 4.5 per cent. alloy, in which the carbon exists in the form of graphite.

With regard to the elongation and reduction of area, the results in general confirm those obtained by Hadfield, Baker, and Paglianti, con-

cerning the effect of silicon (see Figs. 5 and 6). However, the vacuum alloys again show the effect of the lack of carbon in being much tougher in the region of low silicon as well as in the region between 3 and 5 per cent. The latter is particularly significant, as it is in this region that the maximum strength occurs. While the strength maxima for alloys containing small amounts of carbon correspond to zero elongation and reduction of area, the strongest vacuum alloy, in the forged condition, has a reduction of area of 8 per cent. and an elongation of 7 per cent.

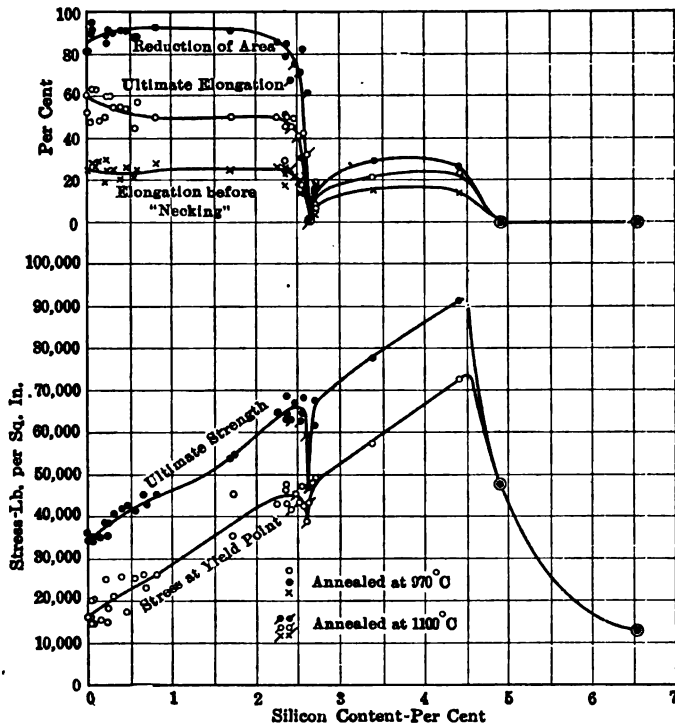


FIG. 4.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS. MELTED IN VACUO. ANNEALED.

In the annealed state the corresponding figures for the same alloy are 24 and 22 per cent.

A very interesting feature, as shown by the curves of Figs. 3 and 4, is the critical point that occurs with about 2.60 per cent. silicon, the characteristics of the alloys in this region being their comparative brittleness. This point was first noticed by the fact that two ingots, containing 2.55 and 2.57 per cent. silicon, respectively, were not forgeable but fell into a mass of crystals that apparently had no adhesive strength. One of these alloys is shown in Fig. 15. That this occurrence was no accident is shown by the fact that the two alloys were made at different times, and they were subjected to forging on different days, in company with other

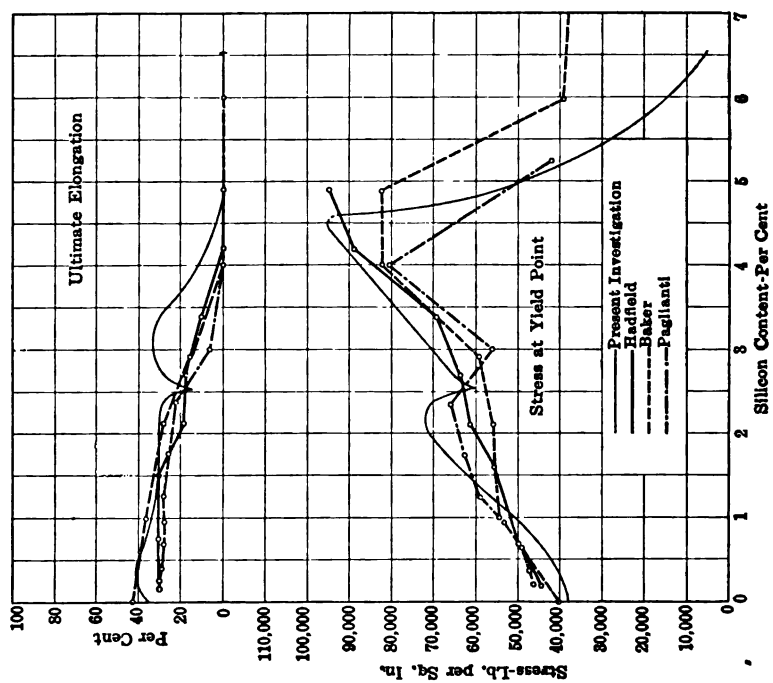


FIG. 5A.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. AS FORGED.

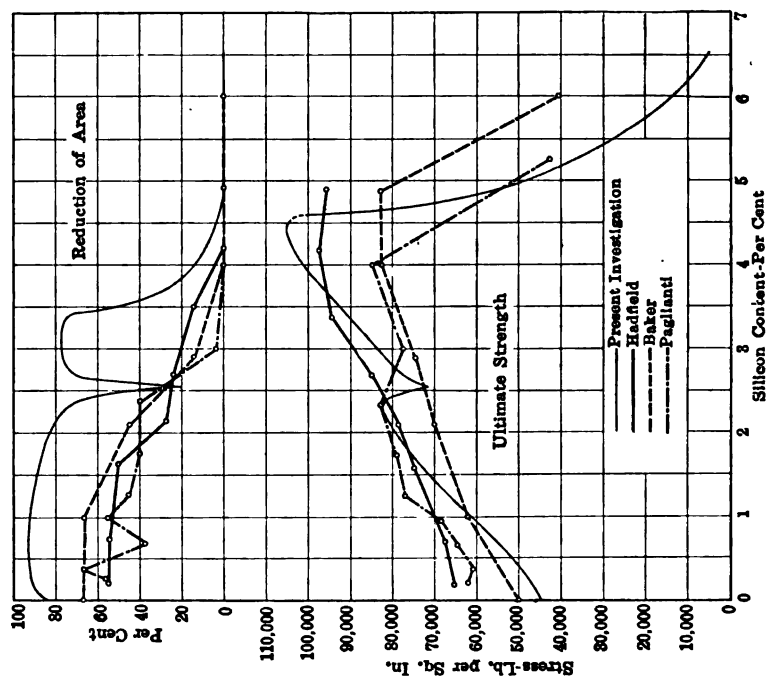


FIG. 5B.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. AS FORGED.

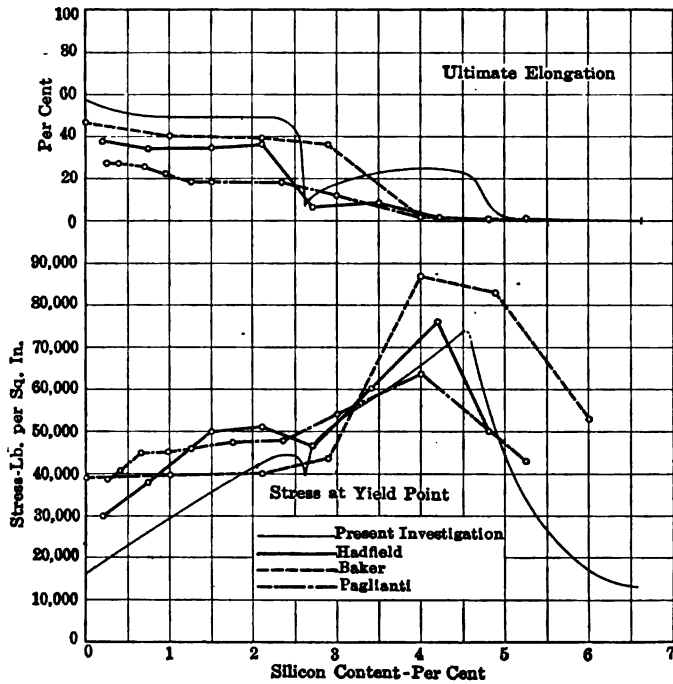


FIG. 6A.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED.

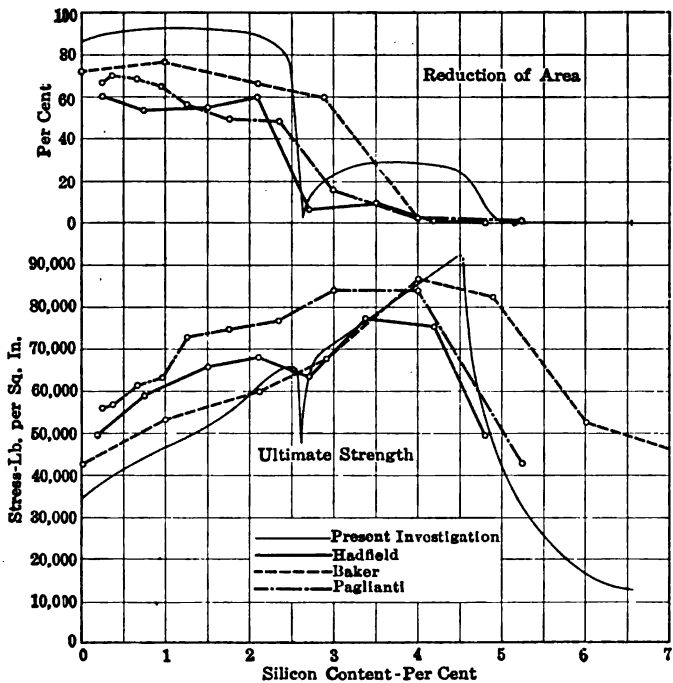


FIG. 6B.—MECHANICAL PROPERTIES OF IRON-SILICON ALLOYS ACCORDING TO VARIOUS INVESTIGATORS. ANNEALED.

alloys that forged perfectly. The structure of the two alloys is identical, consisting of large allotriomorphic crystals $\frac{1}{8}$ in. (3 mm.) to $\frac{1}{4}$ in. (6 mm.) across. Since his first report¹¹ upon the iron-silicon alloys, where this critical point was mentioned, the author has obtained additional experimental data in the region in which this point occurs. This data

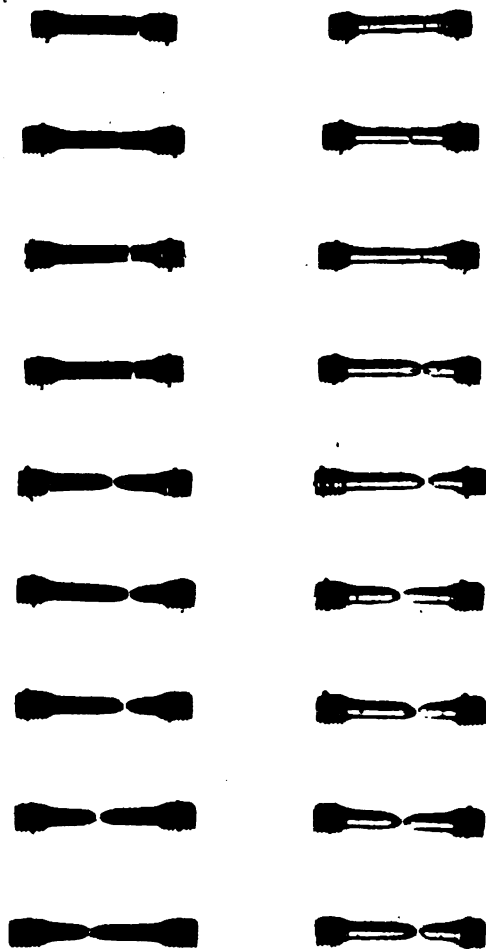


FIG. 7.—SOME OF THE MECHANICAL TEST PIECES AFTER BEING TESTED.

is included in the present report, making it possible to draw definite curves through the critical range. The critical point appears in all the characteristic curves for the mechanical properties both for the unannealed and for the annealed alloys. For the unannealed specimens

¹¹ *Proceedings of the American Institute of Electrical Engineering*, vol. xxxiv, p. 2455 (October, 1915).

there is a sudden drop in the reduction of area and ultimate elongation at 2.25 per cent. silicon, the curves reaching their minima at about 2.56 per cent., which is the silicon content of the two non-forgeable alloys. With increasing silicon contents these curves again rise as suddenly as they previously dropped, attaining about the same values as before the critical point was reached. At 3.50 per cent. the second and final drop begins for these curves, reaching zero at about 5 per cent. silicon.

For the annealed specimens the critical point is even more marked, but the reduction of area and the elongation do not recover as completely after the critical point is passed as was the case with the unannealed specimens. The critical point for the annealed alloys occurs with a silicon content of about 2.65 per cent., slightly higher than for the unannealed alloys.

TABLE IV.—*Results of Magnetic and Electrical Tests, Rods Annealed at 900°C.*

Rod No.	Silicon Content, Per Cent.	Maximum Permeability	Density for Permeability, Gauss	Permeability for $B = 10,000$	Hysteresis Loss, Ergs / C.c. / Cycle		Retentivity, Gauss		Coercive Force, Gilberts / Cm.		Spec. Elec. Res. at 20°C., Microhms	Remarks
					For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$	For $B_{max} = 10,000$	For $B_{max} = 15,000$		
3-54	0.001	23,100	8,500	21,800	764.0	1,610.0	9,400	14,200	0.25	0.30	9.85	
3-55	0.001	22,500	11,000	21,300	875.0	1,790.0	9,100	13,800	0.29	0.36	9.82	
3 Si 16	0.010	25,000	10,000	25,000	795.0	1,770.0	9,480	14,600	0.28	0.35	9.89	
3 Si 15	0.064	22,800	10,000	21,700	782.0	1,738.0	9,100	14,500	0.26	0.35	10.65	
3 Si 05	0.068	37,500	9,000	36,300	405.0	1,210.5	9,480	14,520	0.12	0.23	10.75	
3 Si 06	0.148	47,000	8,000	42,500	398.3	965.0	9,300	14,100	0.12	0.19	11.80	
3 Si 17	0.230	30,000	6,000	26,300	496.0	1,311.0	9,000	13,600	0.15	0.23	11.64	
3 Si 10	0.472	14,000	7,000	12,700	960.0	1,863.0	8,900	11,700	0.30	0.33	16.20	Rod strained in testing.
3 Si 14	0.822	13,500	9,000	13,300	1,215.0	2,432.0	9,100	12,400	0.42	0.53	21.3	Rod strained in testing.
3 Si 31	1.71	18,000	7,000	15,900	800.0	1,541.0	8,400	11,540	0.26	0.32	33.2	
3 Si 18	1.741	14,300	8,000	14,100	935.0	2,162.0	9,000	12,600	0.29	0.42	31.2	
3 Si 27	2.73	16,800	6,000	13,300	821.0	1,779.0	8,270	10,500	0.27	0.34	41.8	
3 Si 25	3.40	20,000	6,000	15,900	560.0	1,390.0	7,900	9,570	0.20	0.23	48.5	
3 Si 36	3.55	14,000	4,500	8,850	802.5	1,812.0	6,900	8,100	0.28	0.34	51.5	Rods contaminated by impure rods.
3 Si 37	4.39	12,750	4,500	7,500	846.0	1,956.0	6,600	7,700	0.29	0.31	58.8	
3 Si 28	4.44	16,100	4,800	10,100	623.0	1,575.0	7,000	8,370	0.16	0.24	57.7	
3 Si 29	4.92	9,100	4,500	5,330	776.0	2,006.0	4,500	5,300	0.27	0.36	66.5	

The author knows of no satisfactory explanation of this critical point, nor has he been able to find any such phenomenon reported by anybody else. From Figs. 5 and 6 it is seen, however, that both Hadfield and Paglianti, and to some extent, Baker, indicate irregularities in some of their curves for a silicon content between 2 and 3 per cent. None of the above investigators, however, and, as far as the author has been able to find out, no other investigator, has recorded in the literature the properties of an iron-silicon alloy containing between 2.50 and 2.67 per cent. silicon.

As critical points are usually associated with the formation of definite compounds of the elements present, it may be of some interest to note that a compound of the formula Fe_{19}Si would contain 2.56 per cent. silicon, and similarly that a compound of the formula $\text{Fe}_{19}\text{Si}_2$ would contain 4.99 per cent. silicon. It was stated above that the first critical point in the present case occurs with a silicon content of 2.56 per cent., and also that there is a sudden change at about 5 per cent. silicon. Whether this agreement is a mere coincidence, or whether these compounds, or others, actually exist, the author is not prepared to say, as conclusive evidence in the way of cooling curves for these particular alloys is not available.

TABLE V.—*Results of Magnetic and Electrical Tests, Rods Annealed at 1,100°C.*

Rod No.	Silicon Content, Per Cent.	Maximum Permeability	Density for Permeability, Gauss	Permeability for $B = 10,000$	Hysteresis Loss, Ergs / C.c. / Cycle		Retentivity, Gauss		Coercive Force, Gilberts / Cm.		Spec. Elec. Res. at 20°C., Microhms	Remarks
					For $B_{\text{max}} = 10,000$	For $B_{\text{max}} = 15,000$	For $B_{\text{max}} = 10,000$	For $B_{\text{max}} = 15,000$	For $B_{\text{max}} = 10,000$	For $B_{\text{max}} = 15,000$		
3-54	0.001	22,800	8,000	21,300	665.0	1,860.0	9,300	13,300	0.20	0.24	9.84	
3-55	0.001	25,800	9,000	25,600	707.0	1,451.0	9,300	12,700	0.23	0.28	9.85	
3 Si 16	0.010	29,000	9,000	28,670	707.0	1,604.0	9,600	14,300	0.21	0.31	9.9	
3 Si 21	0.048	27,000	10,000	27,000	700.0	1,660.0	9,440	14,480	0.23	0.32	10.5	
3 Si 15.	0.064	36,800	9,000	36,300	502.5	1,336.0	9,500	14,300	0.16	0.25	10.67	
3 Si 05	0.068	44,200	9,000	43,500	407.0	1,214.5	9,480	14,200	0.13	0.225	10.78	
3 Si 22	0.091	45,250	9,000	43,500	394.0	929.0	9,500	14,300	0.13	0.17	10.96	
3 Si 06	0.148	66,500	6,500	41,700	286.0	916.0	9,080	12,000	0.09	0.165	11.80	
3 Si 23	0.205	30,200	9,000	29,500	649.0	1,526.0	9,300	14,480	0.20	0.27	12.5	
3 Si 17	0.230											Contaminated by impure rods.
3 Si 07	0.242	36,500	7,500	33,000	436.0	1,346.0	9,700	14,500	0.13	0.21	13.4	
3 Si 08	0.309	44,800	9,000	43,500	445.0	1,412.0	9,600	14,500	0.13	0.24	14.4	
3 Si 09	0.400	22,500	9,000	22,000	725.0	1,820.0	9,440	14,480	0.21	0.32	15.3	
3 Si 10	0.472	31,150	6,200	25,000	535.0	1,358.0	9,300	14,200	0.16	0.21	16.57	
3 Si 11	0.563	25,000	9,000	25,000	601.5	1,624.0	9,200	14,320	0.20	0.28	17.50	
3 Si 12	0.673	28,000	7,000	24,500	468.0	1,636.0	9,200	13,670	0.13	0.23	19.10	
3 Si 13	0.698	20,350	8,000	19,600	780.0	2,220.0	9,300	14,400	0.25	0.40	19.60	
3 Si 14	0.822	30,800	9,500	30,300	542.0	1,765.0	9,200	14,100	0.18	0.35	21.25	
3 Si 31	1.710	30,150	6,500	24,700	440.0	1,292.0	8,700	12,000	0.12	0.22	33.25	
3 Si 18	1.741	33,000	7,000	26,300	416.0	1,112.0	9,200	12,600	0.13	0.19	31.00	
3 Si 27	2.730	46,800	9,500	46,000	404.0	1,260.0	9,100	13,300	0.13	0.23	42.00	
3 Si 25	3.400	63,300	6,500	46,500	280.0	1,025.0	9,100	12,400	0.08	0.15	48.50	
3 Si 36	3.550	36,000	7,500	29,500	419.0	1,157.0	8,920	12,000	0.13	0.21	48.50	Rods annealed in nitrogen.
3 Si 37	4.390	25,700	6,000	15,400	591.0	1,819.0	8,300	10,200	0.20	0.25	56.10	
3 Si 28	4.440	30,200	3,000	15,900	405.0	1,171.0	7,000	8,000	0.12	0.15	57.40	
3 Si 29	4.920	12,200	5,000	7,040	780.0	2,620.0	6,300	7,100	0.26	0.35	66.20	

3. Magnetic and Electrical Properties

The results of the magnetic and electrical tests are shown in Tables IV and V and in Figs. 8 and 9. The latter show at a glance the magnetic and electrical properties of the series, Fig. 8 after annealing at 900°C.,

and Fig. 9 after annealing at 1,100°C. Two maxima appear very distinctly in the curves for maximum permeability, corresponding to two minima in the curves for hysteresis loss and coercive force. The first of these points occurs at a silicon content of about 0.15 per cent., and the second at a silicon content of about 3.5 per cent.

That a maximum—or minimum—should occur for a low silicon content was not surprising in view of the results previously obtained with

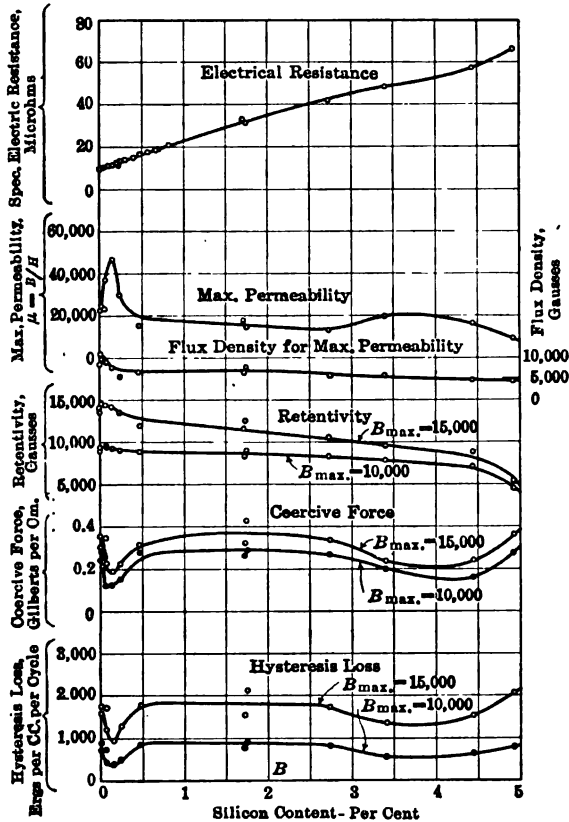


FIG. 8.—MAGNETIC AND ELECTRICAL PROPERTIES OF IRON-SILICON ALLOYS. MELTED IN VACUO. ANNEALED AT 900°C.

pure iron, iron-carbon and iron-boron alloys. In the latter case a maximum was obtained with a trace of boron, evidently on account of a slight purification of the iron, but as soon as the boron content became measurable, the magnetic properties immediately depreciated. The first maximum in the present case can, no doubt, be accounted for in the same way, and consequently this point may be regarded as characteristic of the purest iron obtainable under the present conditions, containing 0.15 per cent. silicon and a small amount of oxygen in the form of iron oxide.

The second maximum—or minimum—was wholly unexpected, as strength and brittleness are not generally associated with high magnetic quality. It is true that previous investigators¹² have found a maximum between 2.5 and 4.0 per cent. silicon, but this was thought to be due to

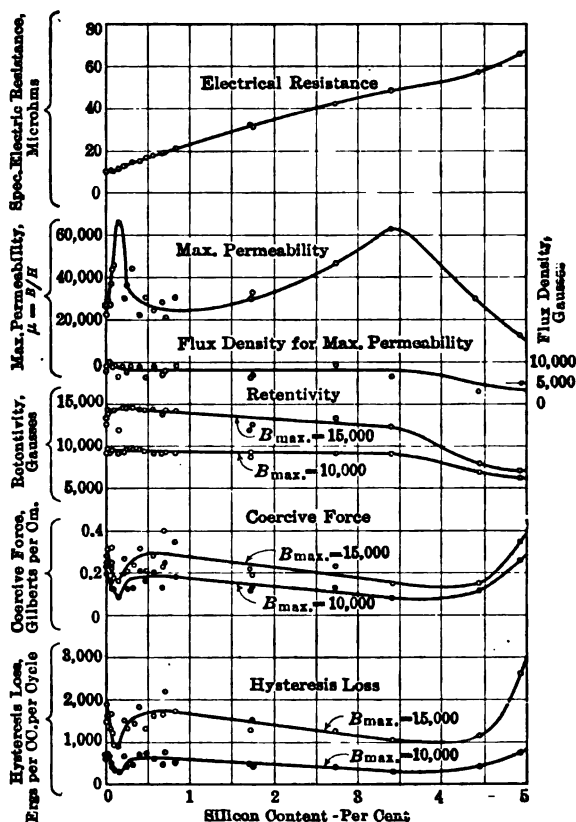


FIG. 9.—MAGNETIC AND ELECTRICAL PROPERTIES OF IRON ALLOYS. MELTED IN VACUO. ANNEALED AT 1,100°C.

the neutralizing effect of the silicon upon the relatively large amounts of impurities, chiefly carbon, present in the iron. In the present case

¹² Barrett, Brown, and Hadfield: *Scientific Transactions of the Royal Dublin Society*, vol. vii, Series II, Part IV, p. 67 (January, 1900). *Journal of the Institution of Electrical Engineers*, vol. xxxi, p. 674 (1901-02).

Gumlich and Schmidt: *Elektrotechnische Zeitschrift*, vol. xxii, p. 691 (1901). Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903); *Journal of the Institution of Electrical Engineers*, vol. xxxiv, p. 498 (1904-05).

Burgess and Aston: *Metallurgical and Chemical Engineering*, vol. viii, p. 131 (March, 1910).

Gumlich and Boerens: *Transactions of the Faraday Society*, vol. viii, p. 98 (1912).

Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

the alloys contained only about 0.01 per cent. of carbon. Thus it seems improbable that the second maximum in this case can be attributed solely to the conversion of 0.01 per cent. of combined carbon into graphite. It seems more probable that the improvements are due partly to this conversion and partly to the complete reduction of iron oxide. According to those hypotheses, the first maximum is due to pure iron in spite of small amounts of iron oxide and combined carbon, while the second maximum is due to pure iron in spite of a relatively large amount of dissolved silicon. If none of the above hypotheses is correct, the only other explanation remaining is that the second maximum is due directly to silicon dissolved in the iron. As an argument against such a theory, Hadfield and Hopkinson¹³ in 1900, and Gumlich¹⁴ in 1912, brought out the fact that silicon reduces the saturation value of iron in direct proportion to the silicon dissolved in the iron, and consequently it did not seem probable that silicon could directly improve the permeability at lower densities. However, it is a curious coincidence that at the same meeting of the Faraday Society at which Dr. Gumlich made the above statement, Dr. P. Weiss¹⁵ read a paper on iron-cobalt alloys, showing that the alloy Fe_2Co has a saturation value 10 per cent. higher than that of pure iron. The author,¹⁶ in coöperation with Dr. E. H. Williams,¹⁷ has shown that while the iron-cobalt alloy Fe_2Co , melted *in vacuo*, has a saturation value 13 per cent. higher than that of pure iron melted under identical conditions, its permeability at low densities is much lower. Evidently both the high saturation value and the comparatively low permeability at low densities of the Fe_2Co alloy must be due to the combination between iron and cobalt, or, in other words, must be attributed directly to the cobalt. There is no reason, then, why the low saturation value and the high permeability at low densities of the 3.40 per cent. iron-silicon alloy cannot both be due directly to silicon. That is, no foundation exists any longer for assuming that there is a direct connection between the saturation value of a certain alloy and its properties at low and medium densities, and it is consequently possible that the second maximum occurring in the maximum permeability curve for the iron-silicon series may be due directly to silicon. More experimental evidence is needed, however, before it is safe to make a definite statement in this respect.

While silicon is thus, directly or indirectly, engaged in improving the magnetic properties of iron, it serves a very useful purpose by enor-

¹³ *Journal of the Institution of Electrical Engineers*, vol. xlvi, p. 225 (1911).

¹⁴ *Transactions of the Faraday Society*, vol. viii, p. 109 (1912).

¹⁵ *Transactions of the Faraday Society*, vol. viii, p. 149 (1912).

¹⁶ *General Electric Review*, vol. xviii, p. 881 (September, 1915).

Proceedings of the American Institute of Electrical Engineers, vol. xxxiv, No. 10, p. 2463 (October, 1915).

¹⁷ *Physical Review*, vol. vi, p. 404 (1915).

mously increasing the electrical resistance of iron, giving the iron the exact characteristics desired for electromagnetic machinery.

The iron-silicon series thus offers two important alloys for electrical purposes, both having high permeability and low hysteresis loss, but differing in that one has a very low, while the other has a very high, electrical resistance.

The values obtained are, undoubtedly, without precedent in the annals

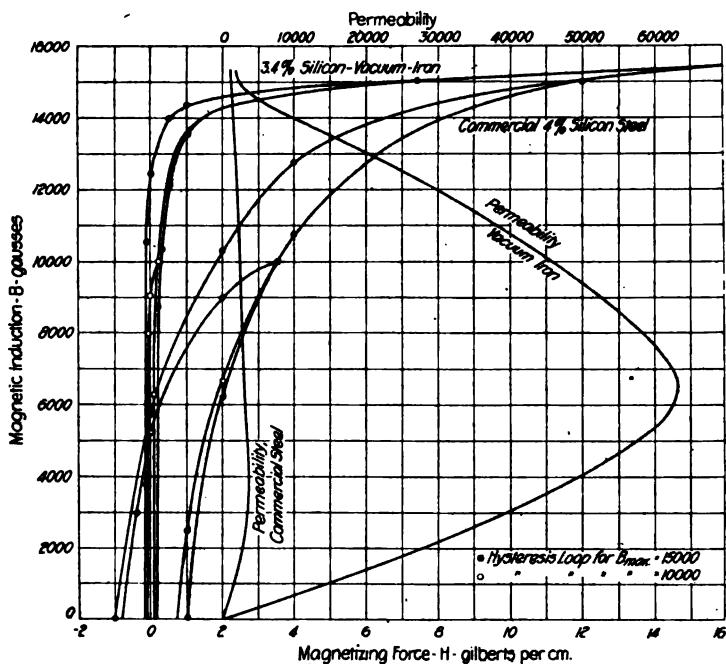


FIG. 10.—COMPARISON BETWEEN VACUUM IRON AND COMMERCIAL STEEL, BOTH CONTAINING BETWEEN 3 AND 4 PER CENT. SILICON. THOROUGHLY ANNEALED.

	Vacuum Iron	Commercial Steel
Hysteresis Loss for $B_{max.} = 10,000$ -ergs. per c.c. per cycle.	280.00	2160.00
Hysteresis Loss for $B_{max.} = 15,000$ -ergs. per c.c. per cycle.	1025.00	4290.00
Specified Electrical Resistance-microhms.	48.50	51.15

of the magnetic properties of iron and iron alloys, and it is only after a careful analysis of the apparatus used and the methods employed in testing that the author feels justified in publishing them. As an extra precaution, however, he wishes to repeat the statement made under "Magnetic Testing" that experimental evidence seems to point toward a larger percentage error due to the compensating current than theoretical con-

siderations would lead to.¹⁸ But even if the maximum error in the results as given should amount to 20 per cent., their significance would not be altered appreciably. Whether the true maximum permeability obtained is 66,500 or 53,200 is of little consequence at the present time, as long as it is reasonably certain that it lies in this neighborhood.

Fig. 10 illustrates the difference in magnetic quality between a silicon-vacuum iron and a commercial silicon steel, both containing approximately the same amount of silicon. The maximum permeability is as 20 to 1, the hysteresis loss, for $B_{max} = 10,000$, as 8 to 1, and for $B_{max} = 15,000$, as 4 to 1, in favor of the vacuum alloy. For $B_{max} = 15,000$ the permeability of the latter is twice the permeability of the commercial iron. The electrical resistance is nearly the same for both alloys, or about five times that of pure iron.

4. Photomicrographs

In the following pages a number of photomicrographs representative of the alloys tested are reproduced. They are arranged in order of their silicon content, the pure iron appearing first and the highest silicon alloy last. With only a few exceptions the "as forged" condition occupies the upper part of the pages, the 900° annealed condition the middle part, and the 1,100° annealed condition the lower part. The magnifications used are either 40 diameters or 10 diameters. In one case recourse was had to 7 diameters. Even with the information thus at hand, exact definite conclusions cannot be made with regard to each alloy taken separately, as the specimen may not in every case be strictly representative of the particular alloy from which it was taken. Furthermore, in comparing the photomicrographs after the different heat treatments, it should be remembered that the specimens had to be repolished after each treatment, and the photomicrographs do not, therefore, represent the same spot in each case. However, general conclusions may be drawn by considering the series as a whole.

From these photomicrographs it is seen very clearly that the iron-silicon alloys, in the range here investigated, consist of only one kind of crystals, thus confirming the results obtained by previous investigators¹⁹

¹⁸ While the experiments in regard to this matter are not yet complete, it is safe to say that the values as given in the paper should be modified to a certain extent. Thus the permeabilities as given are generally too high, as are also the values for residual magnetism and coercive force. While the hysteresis loss for $B_{max} = 10,000$ is too low, the loss for $B_{max} = 15,000$ is too high.

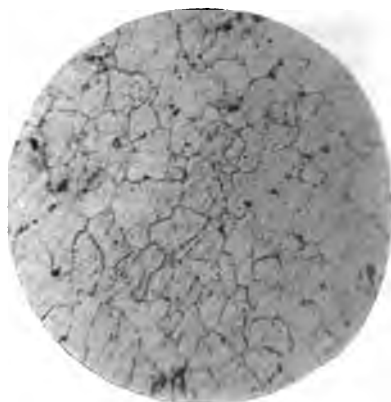
¹⁹ Osmond: *Journal of the Iron and Steel Institute*, vol. xxxvii, p. 62 (1890).

Arnold: *Journal of the Iron and Steel Institute*, vol. xlv, p. 107 (1894).

Baker: *Journal of the Iron and Steel Institute*, vol. lxiv, p. 312 (1903).

Guertler and Tammann: *Zeitschrift für anorganische chemie*, vol. xlvii, p. 165 (1905); vol. lix, p. 384 (1908).

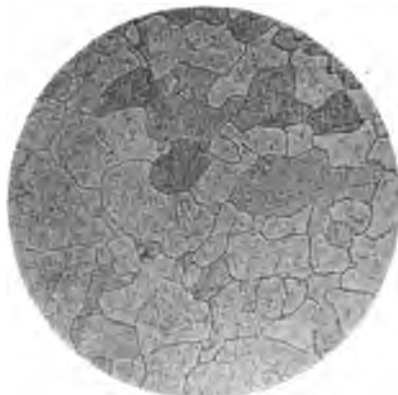
Gontermann: *Journal of the Iron and Steel Institute*, vol. lxxxiii, p. 431 (1911).



As Forged.

FIG. 11A.— $\times 40$ DIAM. PICRIC ACID.

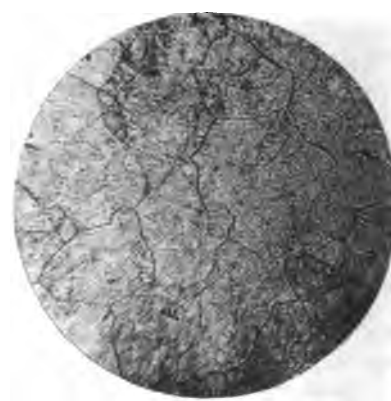
FIG. 12A.— $\times 40$ DIAM. PICRIC ACID.



Annealed at 900°.

FIG. 11B.— $\times 40$ DIAM. PICRIC ACID.

FIG. 12B.— $\times 40$ DIAM. PICRIC ACID.



Annealed at 1100°.

FIG. 11C.— $\times 40$ Diam. PICRIC ACID.

FIG. 12C.— $\times 40$ DIAM. PICRIC ACID.

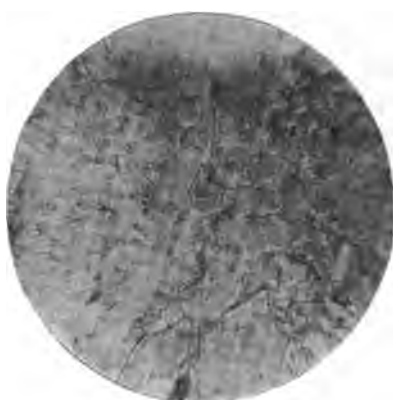
ALLOY No. 3-39. 0.001 PER
CENT. SILICON.

ALLOY No. 3 Si 06. 0.148 PER CENT.
SILICON.



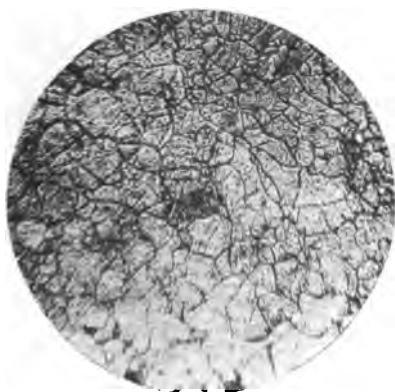
As Forged.

FIG. 13A.— $\times 40$ DIAM. PICRIC ACID. FIG. 14A.— $\times 10$ DIAM. PICRIC ACID.



Annealed at 900°.

FIG. 13B.— $\times 40$ DIAM. PICRIC ACID. FIG. 14B.— $\times 10$ DIAM. PICRIC ACID.



Annealed at 1000°.

FIG. 13C.— $\times 40$ DIAM. PICRIC ACID. FIG. 14C.— $\times 10$ DIAM. PICRIC ACID.

ALLOY No. 3 Si 10. 0.472 PER
CENT. SILICON.

ALLOY No. 3 Si 31. 1.71 PER
CENT. SILICON.



As Cut from Ingot Before Forging.
FIG. 15A.— $\times 10$ DIAM. PICRIC ACID.



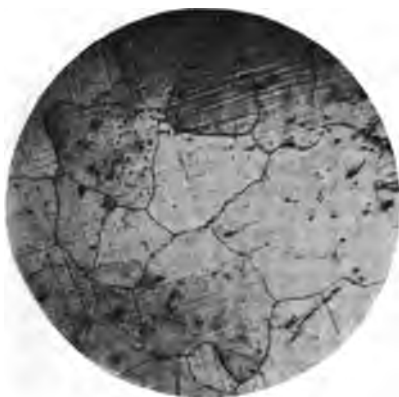
Annealed at 900° .
FIG. 15B.— $\times 10$ DIAM. PICRIC ACID.



Annealed at 1100° .
FIG. 15C.— $\times 10$ DIAM. PICRIC ACID.
NON-FORGEABLE ALLOY NO. 3 SI. 24,
2.57 PER CENT. SILICON.



FIG. 15A2.—NEARLY FULL-SIZE INGOT AFTER FORGING.



As Forged.

FIG. 16.— $\times 40$ DIAM. NITRIC ACID.

FIG. 17A.— $\times 10$ DIAM. PICRIC ACID.



Annealed at 900°.

FIG. 17B.— $\times 10$ DIAM. PICRIC ACID.



Annealed at 1100°.

FIG. 17C.— $\times 7$ DIAM. PICRIC ACID.

SPEC. 1.—ALLOY NO. 3 Si 25.

SPEC. 2.—3.40 PER CENT. SILICON.



As Forged.

FIG. 18A.— $\times 40$ DIAM. PICRIC ACID.

FIG. 19A.— $\times 40$ DIAM. PICRIC ACID.



Annealed at 900°.

FIG. 18B.— $\times 40$ DIAM. PICRIC ACID.

FIG. 19B.— $\times 40$ DIAM. PICRIC ACID
SLIGHTLY REPOLISHED.



Annealed at 1100°.

FIG. 18C.— $\times 40$ DIAM. PICRIC ACID.
WASHED IN 10 PER CENT. HYDROFLUORIC
SOLUTION.

FIG. 19C.— $\times 40$ DIAM. PICRIC ACID.
WASHED IN 10 PER CENT. HYDROFLUORIC
SOLUTION.

ALLOY No. 3 Si 29. 4.92 PER
CENT. SILICON.

ALLOY No. 3 Si 32. 6.57 PER
CENT. SILICON.

that iron and silicon, for silicon contents below about 15 per cent., form a solid solution everywhere between the freezing point and ordinary temperature.

Below 1 per cent., silicon appears to have no marked effect upon the structure of the iron, either as forged or annealed. Annealing at 900°C. does not seem to change the structure of the alloys in this range, but annealing at 1,100° breaks up the large crystals into smaller ones, giving an appearance of a very fine structure.²⁰ There is no sign of foreign substances in these structures, other than those arising from imperfect polishing. After passing the 1 per cent. mark, silicon begins to show its effect. The crystals are generally larger than for the pure iron and are readily polished in relief, showing that they are not of uniform hardness. There is no breaking up of the crystals by the 1,100° annealing as in the case of the low alloys. The 1.71 per cent. alloy, as seen in Fig. 14, shows



FIG. 20.—NON-FORGEABLE ALLOY NO. 3 Si 30. 8.55 PER CENT. SILICON. INGOT AFTER FORGING. NEARLY FULL-SIZE.

a very irregular structure as forged, and after the 900° annealing, but this gives way to a structure of more regularity by the 1,100° annealing. The structure of the non-forgeable alloy is shown in Fig. 15, exhibiting very large, uniform crystals, measuring $\frac{1}{8}$ in. (3 mm.) to $\frac{1}{4}$ in. (6 mm.) across. In the first specimen used for the 3.40 per cent. alloy, the crystals are of about the same size and shape as the ones for the non-forgeable alloy, as seen in Fig. 17. In order to further investigate this singularity, another specimen from the same alloy was prepared, and this showed a much more normal structure (Fig. 16). Specimens for two other alloys

²⁰ This phenomenon is explained by Stead and Carpenter, *Journal of the Iron and Steel Institute* (September, 1913) as follows: Upon heating the iron above the A_r point (900°C. for pure iron) and holding it in the region of the gamma modification sufficiently long, both the alpha crystals and their nuclei will be destroyed, giving place to gamma crystals. If now the iron is cooled numerous alpha nuclei will be formed simultaneously on passing through A_r , and the resulting crystals are consequently small.

See also Oliver W. Stovey: A Microscopic Study of Electrolytic Iron, *Transactions of the American Electrochemical Society*, vol. v, p. 201 (1904).

with very nearly the same silicon content were also investigated, and these also showed structures that were quite normal, so that the large crystals, shown in Fig. 17, are evidently freaks, caused by peculiar conditions. It may be that the specimen was taken at one end of the forged rod. Nevertheless, the occurrence of these enormous crystals is very interesting, not only on account of the enormous size, but also because it shows that the size of the crystals in and of itself does not prevent the material from being forgeable. The remainder of the photomicrographs exhibit quite normal structures with no marked change caused by the two heat treatments.

The principal difference between these photomicrographs and the ones published by previous investigators²¹ is the absence of foreign

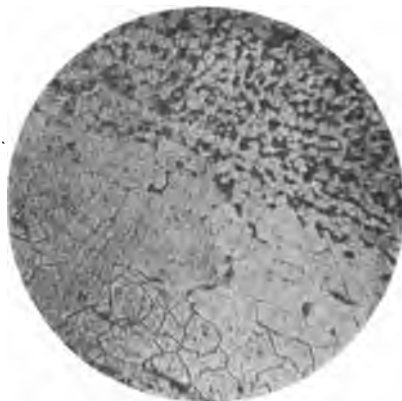


FIG. 21.—SWEDISH CHARCOAL IRON AS CUT FROM PLATE ANNEALED AT 945°C. CARBON CONTENT 0.163 PER CENT. MAGNIFICATION $\times 40$ DIAMETERS.



FIG. 22.—SWEDISH CHARCOAL IRON, AFTER BEING RESMELTED IN VACUO. AS FORGED. CARBON CONTENT 0.008 PER CENT. MAGNIFICATION $\times 40$ DIAMETERS.

matter in the structure of the vacuum alloys. Even Baker, whose alloys contain only 0.04 per cent. carbon, shows in his photomicrographs besides small amounts of pearlite or graphite, some other foreign substance that could not be explained at the time. It seems probable, in view of the method used for melting and the mechanical properties of his alloys, that these other foreign substances may be oxides. The other investigators invariably show comparatively large amounts of pearlite for low alloys and patches or spots of graphite for high alloys. The size of crystals for the vacuum alloys, excluding the abnormal cases, is very much larger than for the less pure alloys. This is true both for low and high silicon contents.

²¹ Guillet: *Révue de Métallurgie*, vol. i, p. 46 (1904).

Gumlich and Goerens: *Transactions of the Faraday Society*, vol. viii, p. 98 (1912).

Paglianti: *Métallurgie*, vol. ix, p. 217 (1912).

IV. SUMMARY AND CONCLUSIONS

The results recorded in the previous pages may be summarized as follows:

1. By means of the vacuum method of melting it is possible to obtain a decidedly purer product than has thus far been obtained in any other manner. Consequently by the use of this method more definite conclusions, than have hitherto been possible, can be drawn with regard to the effect of silicon upon iron.

2. Silicon, like boron, has a double effect upon iron. Part of it combines with the iron and remains in solid solution throughout the cooling of the alloy, while a smaller part reduces the iron oxide present.

3. The tensile strength of the vacuum product follows in general the same law as alloys made under ordinary conditions, but the ductility of the former is much greater, particularly below 2 per cent. and above 3 per cent., probably due to the absence of carbon. The maximum tensile strength of 105,000 lb. per square inch (73.5 kg. per square millimeter) occurs with a silicon content of 4.5 per cent.

4 The limit of forgeability lies between 7 and 8 per cent. silicon. A critical range occurs between 2.50 and 2.60 per cent., in which the alloys are exceedingly brittle, in some cases being not even forgeable. The chemical composition of the latter corresponds to a compound of the formula Fe_{19}Si .

5. With regard to the magnetic properties, the vacuum alloys exhibit most remarkable characteristics. The best alloys are obtained with about 0.15 per cent. and 3.40 per cent. silicon after annealing at $1,100^{\circ}\text{C}$. The maximum permeability for both of these alloys is above 50,000, and the hysteresis loss for $B_{\text{max}} = 10,000$ and 15,000 is about 300 and 1,000 ergs per cubic centimeter per cycle respectively. This hysteresis loss is one-eighth and one-third of the corresponding loss for commercial silicon steel. The most favorable annealing temperature is in every case $1,100^{\circ}\text{C}$.

6. The specific electrical resistance increases about 13 microhms for the first per cent. silicon added. For each additional per cent. added the increase is about 11 microhms. Consequently the 3.40 per cent. alloy mentioned under 5 has a resistance nearly five times that of the 0.15 per cent. alloy.

By the vacuum process two silicon alloys have thus been produced that have very valuable characteristics; one low in silicon, not very strong, but extremely ductile, of high permeability, low hysteresis loss, and of low electrical resistance; the other high in silicon, very strong, moderately tough, of high permeability, low hysteresis loss and of high electrical resistance. The properties for these two alloys are summarized in Table VI. The first is evidently suitable for use in places where high permeability and low hysteresis loss are the chief requirements, while

the second alloy is suitable for electromagnetic machinery, principally transformers, where a low eddy-current loss is an additional requirement.

TABLE VI.—*Properties of the Two Best Iron-Silicon Vacuum Alloys*

Silicon Content, Per Cent.	Stress at Yield Point, Lb. per Sq. In.	Ulti- mate Strength, Lb. per Sq. In.	Elonga- tion, Per Cent.	Reduction of Area, Per Cent.	Maximum Permeability	Density for Max. Perme- ability, Gausses	Hysteresis Loss, Ergs/ c.c./Cycle		Spec. Elect. Resist- ance, Mi- crohms
							For $B_{max}=10,000$	For $B_{max}=15,000$	
0.15	18,500	37,000	56	90.0	above 50,000	6,500	286 ¹	916 ²	11.80
3.40	58,000	76,500	21	28.5	above 50,000	6,500	280 ¹	1,025 ²	48.50

¹From data recently obtained with rings these values may be 10 to 20 per cent. low.

²From data recently obtained with rings these values may be 5 to 10 per cent. high.

The mechanical properties of the second alloy make possible its use also in dynamo machinery, where the present commercial silicon steel cannot be used on account of its brittleness.

It should be pointed out that electrolytic iron is not essential to the attainment of high magnetic quality. Any low-carbon iron that is practically free from phosphorus, sulphur and manganese, *when melted in vacuo*, will give magnetic properties approaching very closely to those obtainable with electrolytic iron.²² At the present time there are obtainable a number of commercial grades of iron that come within these specifications (see Figs. 21 and 22).

The author fully realizes the difficulties that have to be overcome before the vacuum iron can be used in the industries in competition with materials available at the present time. Suggestions have, nevertheless, already come from several sources for its employment in places where its high cost of production is not of vital importance, and there is no doubt but that its usefulness in limited fields will increase as time goes on. However, the author would like to suggest that the results here given be partly considered as a new indication of the possibilities obtainable in the realm of magnetic properties, rather than as the final word regarding certain properties of iron-silicon alloys that can be turned into commercial use at the present time.

In conclusion the author wishes to acknowledge his indebtedness to many persons in various departments of the University of Illinois who have given their time and thought to the investigation. In particular, he wishes to mention Prof. E. B. Paine, acting head of the Electrical Engineering Department, W. A. Gatward, Fellow in the Engineering Experiment Station, for his conscientious work in connection with the magnetic testing, and Messrs. J. M. Lindgren and F. H. Whittum of the Chemistry Department for the chemical analyses.

²² Bulletin No. 72, *Engineering Experiment Station, Illinois University*, pp. 33, 44 (1913-14).

Zinc-Dust Precipitation Tests

Discussion of the paper of NATHANIEL HERZ, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 104, August, 1915, pp. 1507 to 1513.

GILBERT RIGG, New York, N. Y. (communication to the Secretary*).—The following statement by Mr. Herz, if not properly qualified, is apt to be misleading:

"The nature of alloys of cadmium and lead with zinc has not been studied by modern metallographic methods." This statement is quite true so far as the ternary system Pb-Cd-Zn is concerned, but the binary systems Pb-Cd, Zn-Cd, and Pb-Zn have been studied by several investigators. Furthermore, sufficient metallographic work has been done on spelter itself to give us a very fair knowledge of the high zinc corner of the Zn-Cd-Pb diagram. A list of references is given at the end of this discussion.

Mr. Herz continues: "It is known that lead is slightly soluble in zinc, and on cooling, about 1.3 per cent. of lead is left in the zinc as a solid solution, any more lead segregating as a solution of zinc in lead. Therefore, if cadmium is absent the metallic portion of the zinc may contain 1.3 per cent. of lead solution, possibly in molecular form, while any excess must be in larger particles. This can be shown by preparing an alloy and cooling it slowly, cutting out all parts showing segregations of lead, and analyzing the remainder."

Now it is quite true that in a bath of lead and zinc kept only slightly above the melting point of the latter, the upper (zinc) layer will contain approximately the amount of lead stated by Mr. Herz (about 1 per cent.). As the temperature of the bath increases, the upper layer—the solution of lead in zinc—will become increasingly richer in lead as Spring and Romanoff have shown. With falling temperature, however, the solubility of lead in zinc rapidly diminishes. Arnemann, and Heycock and Neville have placed the limit of solubility of lead in zinc at a temperature of 418°C. at 0.5 per cent. lead. At this temperature, of course, the mixture begins to freeze. Doubtless the reason that we are unable to reduce the lead content of spelter below about 1 per cent. by liquation methods is due to the fact that at the freezing temperature the metal is somewhat viscous, and holds the lead which has been thrown out of solution (above about 0.5 per cent.) in the form of an emulsion—a suspension of minute globules. Microscopic examination of spelters confirms this. Spelters containing from 1 to 1.5 per cent. lead under higher powers of the microscope show the lead as distinct globules. If Mr. Herz will prepare the "unsegregated" parts of his lead-zinc alloy, etch with nitric acid, and examine under the microscope, he will be quite convinced that the 1.3 per cent. of lead is not present as a solid solution.

Mr. Herz also states: "Cadmium is soluble in melted zinc in all proportions and insoluble in solid zinc, but separating only at the instant of solidification does not segregate in large masses." Such investigations

* Received Dec. 29, 1915.

as have been published are rather conflicting regarding the presence or absence of solid solutions of cadmium in zinc; some investigators claim that they are absent, while others indicate their presence to a limited extent. My microscopic examination of spelters containing cadmium has invariably shown the presence of solid solutions. On the other hand, with very slow cooling the cadmium separates as a definite constituent. This, of course, only applies to cadmium less than 1 per cent. In other words, in the stable system, produced by slow cooling, solid solutions are absent; but in the metastable system, which is practically the universal case with spelters containing cadmium, solid solutions are present due to the rapid cooling.

Mr. Herz's statement: "The probability is that cadmium would increase the solubility of lead in solid as well as melted zinc," possibly is true. Under the microscope, lead is not as easily discerned in spelters containing considerable amounts of cadmium as in those free from it.

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¹ The references under the heading Cd-Pb are incomplete. The article referred to, however, gives full data on the system.

Kick vs. Rittinger

Discussion of the paper of A. O. GATES, presented at the San Francisco meeting, September, 1915, and printed in *Bulletin* No. 105, September, 1915, pp. 2023 to 2051.

ALGERNON DEL MAR, Los Angeles, Cal. (communication to the Secretary).—Since reading the first discussion of the work done in crushing, I was impressed with the possibilities inherent in any law that would give us the means of comparing different machines crushing the same ore. It is manifestly impossible to compare two machines crushing different ores unless the specific resistances of the ores are known, and it is likewise inadequate to compare two machines crushing the same ore through radically different meshes, for the resistance to fracture varies in the same ore with the degree of fineness, so for this particular phase of comparison we must experiment on ores crushed through nearly the same range of sizes.

After a diligent study of the literature on the subject I concluded that while no exact law could be formulated to represent the work done in crushing, a close approximation was possible on the assumption that the work done in crushing was proportional to the surface exposed by the crushing force and not to the volume at every stage, and thus I came to uphold Rittinger. Having at my disposal no means to test the accuracy of the two theories by actual experiment, I could only reason from a mathematical standpoint. Stadler and Taggart, both reasoning from the same standpoint, proved to their satisfaction the falsity of the Rittinger theory and the correctness of Kick's law which holds that the work done in crushing is proportional to the volume at every stage. We waited for a practical demonstration to prove the worth of either theory and Professor Bell, at McGill University, sent word that by actual demonstration Kick's law was proved to be correct; to my knowledge the figures were not made public, and now the author of the paper under discussion has proved that Rittinger was correct after all. Taggart laid emphasis on the point that in the Rittinger theory the distance factor in the general formula—work = resistance $\times$ distance—was not taken into account but it will be noticed that Mr. Gates in each experiment has recorded the distance through which the crushing force acts.

In 1914 I attended a meeting of the Institute in New York at which Professor Taggart's paper was presented but not read; there was no discussion. Last spring I read a paper on the same subject before the Los Angeles Section of the Institute, but the discussion drifted off into the degree of fineness to which a rock breaker could be made to grind economically, and I pointed out to the members present that by the use of the Rittinger formula it was possible to determine this limit. I mention these facts here to indicate that unless we are able to show the practical import of a law for crushing, it is impossible to arouse interest in the subject.

It appears strange that while Stadler regards the Rittinger theory as pure nonsense, and Taggart argues that the distance factor is not considered, Gates proves the theory by experiment. Likewise, there is the anomaly that mathematically both laws can be proved correct according to individual ideas of how the crushing forces act.

Although I have lacked the advantage of a physical laboratory in which to make a practical demonstration, I have in a crude way made some experiments which appear to substantiate my original postulate that the work done in crushing is proportional to the reciprocal of the diameters to which the ore is crushed. Stadler in 1910-11 brought forward his energy units (E. U.) as a means of computation; in 1912 I presented the method of computing crushing efficiencies from the reciprocals of the diameters, and in 1913 Gates used his crushing-surface diagrams to demonstrate the same computations. The crude experiments I refer to were made with a small stamp on concrete blocks with an artificial plane of weakness through the center of one face, by which means I demonstrated to my satisfaction that the work done in crushing is proportional to the surface exposed. These experiments are recorded in the paper read before the Los Angeles Section and need not be repeated here. For those who have not a clear conception of the two formulas involved in this discussion it might be well to cite them for comparison.

The Rittinger formula for the mechanical units used in crushing is D/d , and while Stadler gives the formula for his energy units as $-10 \log S$, where S is the diameter of the screen aperture, a simpler formula is $\log D/d$. The Rittinger formula is supposed to represent the work done at the moment of fracture.

As stated by Gates, the Kick formula represents the work done within the elastic limit. We then have some phenomenon at the elastic limit which will account for the difference. The point for future study is here indicated, and the question whether this difference can be accounted for in the work returned by the crushing machine on the release of the crushing force, the heat developed by compression and dissipated at the moment of fracture, or by other physical changes, affords opportunities for original research in this field.

The work done in crushing is generally considered to be an academic subject with no practical application, and only by demonstrating that this is not true can interest in the subject be awakened. At the Winona mill in the Lake Superior district two Hardinge mills, one 6 ft. in diameter and 5 ft. straight face, the other 8 ft. diameter and 30 in. straight face, are run side by side on the same ore of the same degree of fineness. It is desired to know, for future installations, which mill is the more effective. H. H. Seeber, the manager, prepared a chart showing the power consumed by each mill, screen analyses of the feed and discharge, and the tonnage, and from these data it was found that the smaller diameter mill was doing

better work. This shows one practical application of the laws of crushing. Had this demonstration been carried far enough it might have been found, by the same laws, that the size of the feed for the two mills favored the one of less diameter and that by feeding a coarser product the bigger diameter mill would be proved most efficient. This shows another application of the law, *i.e.*, a means of determining the best size of feed or discharge for a crushing machine.

In this connection may be mentioned a recent episode which, though a little removed from the subject under discussion, may be of interest. Stadler has calculated the relative efficiency of the tube mill and stamps on the Rand to be 17 to 70 (omitting fractions). In passing to and fro before the discharge end of a tube mill one is often impressed with the great increase of heat above the atmospheric temperature, and it appeared to me that in some way this conversion of mechanical work into heat might give us another method of comparing efficiencies. For the purpose of obtaining some figures I crudely measured the difference in temperature at the feed and discharge ends of a tube mill working in the district and found it to be 4°C. To determine more fully the bearing of this formation of heat I called upon another mine superintendent and stated my desire to measure the difference in temperature at the two ends of his mill. He gave me permission, but impressed upon me the fact that the subject was purely academic and had no practical import. I found the difference in temperature at the two ends of his mill to be 10°C. and, as might be expected, the differences in temperatures of the two mills bore a close relation to their output. I brought this to the manager's attention, thus clinching my contention that while the subject is no doubt academic, by following its ramifications some interesting and useful hints that have practical application might be found. If all the heat generated in a crushing machine could be measured, knowing the specific heat of the substances heated we might be able to settle Kick vs Rittinger.

We must thank Mr. Gates for his untiring energy in endeavoring to fathom this subject, and while I am convinced of the soundness of his arguments, we should like to see Professor Bell's figures and likewise hear from our own State University on the subject.

H. STADLER, London, S. W., England (communication to the Secretary*).—The laws that the change of stress and change of configuration are proportional, and that the work spent upon producing these stresses is proportional to the *product* of change of stress into change of configuration, are accepted by physicists and technologists all over the world. Expressed as above they are general and refer to all forms in which energy appears (dynamic, kinetic, electric, heat, chemical affinity, etc.) and to all sorts of changes of configuration (elastic deformation, plastic deformation with rupture as the ultimate end stage, displacement, change of

* Received Oct. 16, 1915.

state, etc.). For the sake of convenience these laws are expressed in a different way in each of the various branches of technology, using the terms commonly employed in that particular line. In statics and mechanics, for instance, it is said that the force, or the breaking load, varies as the square of the homologous dimensions; and that the mechanical work is proportional to the *product* of force into the distance through which the force has to move against the opposing stresses. These laws are well understood and are familiar to all engineers. Applied to crushing we may accept Kick's version which correlates the energy required for causing the rupture of a body with the reduction of its volume (or weight). According to the so-called Rittinger theory, which is offered in opposition to the above laws, "work" and "force" vary at the same ratio; advocates of this hypothesis disregard the distance factor. Its real position becomes apparent whenever it is subjected to practical tests.

All branches of technology abound with examples of the universal validity of Kick's law, or of applications of it; we may take it either way. The law holds good in extreme cases, from the coarsest possible crushing by blasting down to the elastic deformation of gases and liquids. The so-called Rittinger's theory, on the other hand, breaks down by simply glancing at a grading analysis, for instance, of the battery pulp given below:

Battery Pulp, 9 Mesh (0.272 in.)

Grading		Kick's Law			Rittinger's Law		
I. M. M.	Weights, Per Cent.	Volume Factor	Efficiency of Stamp		Surface Factor	Efficiency of Stamp	
			A	B		A	B
+ 5	5.0	9	0.450	Reduced to 100 per cent. 15.214	10	0.500	Reduced to 100 per cent. 82.068
+ 8	14.8	11	1.628		16	2.368	
+ 12	12.4	13	1.612		24	2.976	
+ 20	11.1	15	1.665		40	4.440	
+ 30	9.0	17	1.530		60	5.400	
+ 50	9.7	19	1.843		100	9.700	
+ 80	7.5	21	1.575		161	12.075	
+ 120	4.5	23	1.035		238	10.710	
+ 200	4.9	25	1.225	0.0	400	19.600	0.0
- 200	21.1	28	5.908		645	136.095	
100.0			18.471	15.214		203.864	82.068
Ratio:			100	82.37		100	40.25

Assuming for argument's sake that by any means (yet to be discovered) the production of the 21.1 per cent. of the -200 grade could be prevented in a second stamp (with a proportional increase of the percentages on the remaining grades), the volume factor taken as a standard for the

measurement of the energy consumed in the two cases would account for a lower power consumption of 17.6 per cent. for the improved stamp, and the surface factor of 59.8 per cent., which latter figure is obviously absurd.

In his original article Mr. Gates advances a "theoretical proof of the Rittinger theory," claiming that if a cube be ruptured by a shearing force applied by two offset faces, this force would act directly in the vertical fracture plane, "the mass of the cube away from this surface receiving practically no pressure or deformation." The fallacy of this reasoning has been exposed by several writers in leading technical journals, but Mr. Gates reprints his "theoretical proof" *in extenso*.

Another argument much abused by the opponents of Kick's law is the contention that it refers only to elastic, but not to plastic, deformations. These two critical points are, in highly elastic materials such as are generally subject to crushing, so close together that this objection could be discarded as having no appreciable effect on the results, but Kick's law does not need to have recourse to such an allowance, for a simple household sausage machine is quite good enough for demonstrating experimentally that the work spent upon causing the analogous plastic deformations is proportional to the volume, or weight, of the meat paste.

Mr. Gates opens his argument by pointing out that the ratio of the energy expended for breaking a 16-in. cube in two stages, first to 1-in. cubes and then to $\frac{1}{16}$ -in. cubes, is 1:1 for Kick's law and 1:16 for Rittinger's law. With such a clear issue one would have expected that an experimentator would have proceeded, with the excellent apparatus at his disposal, to measure the work required for crushing varying sizes of geometrically and technologically similar bodies of any shape. Instead of carrying out such simple tests, Mr. Gates engaged in experiments made with mixed sizes, an undertaking beset with difficulties which he increased by introducing all sorts of easily avoidable complications.

One of these is the creation of a new unit, the "mesh-gram," which in a rather cumbersome definition is said to mean the area of surface exposed in reducing 1 gram of particles of equal size to a smaller size, this area to be measured by the difference of the reciprocals of the diameters. The reference to weights is utterly superfluous in a case where we have to deal only with relative values for which the reciprocal of the diameter is the natural representative.

Other complications which make a scrutiny of the data presented practically impossible are: The use of different materials, the confused and prolix way of expressing simple facts and relations, the tampering with samples by the occasional and unsystematical screening out of the "fines," the too small scale of the tests in dealing with mixed-sized products of which a sufficient amount should be available for arriving at fair average values, and, last but not least, the complete neglect to take the

precautions necessary to secure the strict fulfillment of the requirement of analogous conditions in all stages of operation.

Under these circumstances it would serve no useful purpose to go into the details of the tests, and I content myself by showing how easily a blunder can be made if the experimenter, undertaking to overthrow a universally recognized law of nature, is under the influence of the prejudice that "no elaborate apparatus and apparently no very high degree of accuracy will be required to prove its fallacy."

No objection can be raised against an attempt to check the merits of the crushing laws by tests approaching practical working conditions, provided that the conditions be in all crushing stages proportionate and in accordance with the premises common to all crushing laws. That this was not the case in the present tests is evidenced by the enormously more rapid increase of pressure required for causing an appreciable compression of the crushing beds, when the finer sizes came to be dealt with. In fact, if no "fines" had been screened out, a point would soon have been reached where even the greatest pressure would have had no more crushing effect. Mr. Gates correctly inferred from this observation that free crushing is more efficient than choke-crushing, but he failed to realize that in his tests choke-crushing took place to a much larger extent in the fine- than in the coarse-crushing stages. The temporary remedy of screening out "fines" at intervals did not prevent choking taking place in the immediately preceding test by the fragments produced during the operation of crushing. The conditions of the tests were obviously not analogous in the series of experiments and this operated to the detriment of the efficiency of the fine-crushing stages. The results are therefore not such as warrant drawing inferences on the correctness of a crushing law which conversely is expected to be the theoretically correct standard for the measurement of the efficiency of crushing appliances under varying working conditions.

The great advance of technological knowledge attained as the result of evolution during centuries of honest and thorough research work, to which great (mostly French) scientists devoted their lifetime, is recorded in an abundant literature, and it is not too much to expect that mining engineers who propose to lecture or write on these matters should make themselves acquainted with it and abandon the practice of drawing their wisdom from antiquated text-books on ore dressing.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 29th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Researches on Fire-Damp

BY ENRIQUE HAUSER,* MADRID, SPAIN

Translated from the French by Bradley Stoughton and G. A. Burrell

(New York Meeting, February, 1916)

FIRE-DAMP is a mixture of methane with other inert gases or combustible gases. The inert gases in question are carbonic acid, water vapor, nitrogen, etc. The combustible gases are hydrogen, ethane, etc. The study of the properties of fire-damp, therefore, becomes a study of the properties of methane and the influence on its properties of the different gases we have mentioned.

Characteristic Properties of Methane

1. *Respirability*.—Methane being a paraffin is characterized by its weak chemical affinities. From the physiological point of view one could substitute it for nitrogen of the air at least momentarily, without experiencing any difficulty in breathing the mixture consisting of 21 per cent. O_2 + 79 per cent. CH_4 . I have tried this on myself, making the gas exhaled from the lungs burn subsequently in a Bunsen burner.¹ From the chemical point of view, this weak affinity permits the gas to be burned in certain circumstances with hydrogen and oxide of carbon in an oxygenated mixture, with the result that the last two gases achieve only a fractional combustion. This practice has been utilized with success in the methods of analysis which I have perfected and which I describe later in this paper.

2. *Retardation of Flashing Point*.—A characteristic property of methane is its retardation of flashing point, noted by Davy in 1816 and studied by Mallard and Le Chatelier in 1883.² Taffanel and Le Floch have recently confirmed these results.³

Let us recall the nature of this retardation:

When a mixture of combustible gases with air or with oxygen is

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¹ El Grisú en las Minas de carbón.—*Primera Conferencia experimental*, 1907. Leçons sur le grisou, *Première Conférence expérimentale*, 1908, pp. 57–58.

² *Annales des Mines*, 1883, pp. 274–296.

³ C. R. de l'Ac de S., 19 Mai, 1913.

submitted to the action of heat, combustion commences and continues slowly without any external manifestation indicating the occurrence of a chemical reaction. But, if we raise the temperature little by little a point is reached where this combustion takes place rapidly, manifested externally in the phenomena of light and finally in the diminution of volume of the mixture. It is the occurrence of these latter phenomena which is called the flashing point of the gas. Thus, according to the experiments of Mallard and Le Chatelier, in a mixture of carbonic oxide and oxygen, combustion commences with appreciable rapidity at a temperature of $400^{\circ}\text{C}.$; at $447^{\circ}\text{C}.$ the proportion of the mixture which

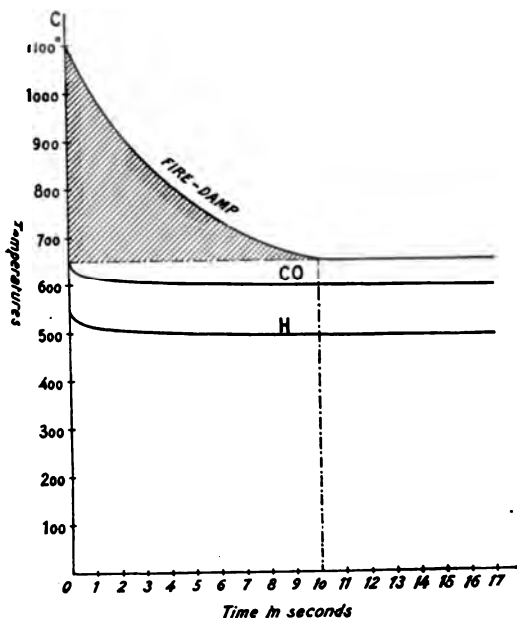


FIG. 1.

burns in a second is 1 in 1,000; at $615^{\circ}\text{C}.$ the proportion is 1.5 in 1,000; and, at $650^{\circ}\text{C}.$, the flashing point is immediately reached. That is to say, this great change occurs in a temperature interval of only 35° . At $650^{\circ}\text{C}.$ the whole mixture is burned in some thousandths of a second.

The same general result is noted in the case of hydrogen: For example, in an experiment made at $540^{\circ}\text{C}.$ only one-half of the mass was burned, while at $555^{\circ}\text{C}.$ combustion was completed on the instant.

In mixtures of methane with air these phenomena of retardation are very much amplified: Slow combustion commences at $450^{\circ}\text{C}.$ without arriving at the flashing point. More rapid combustion occurs at $650^{\circ}\text{C}.$, requiring about 10 sec. to accomplish the burning of the mixture. This

duration, or this retardation, becomes almost negligible at 1,000°C. In mixtures of fire-damp and oxygen, the retardation of the flashing point is a little less.

In order to represent these phenomena better I have drawn a curve (Fig. 1) where the ordinates represent temperatures and the abscissæ represent the duration in seconds. The value of the abscissæ will then indicate to us the heat required for the gas in question to reach the flashing point at any given temperature. In the figure herewith are represented the curves of hydrogen, carbonic oxide, and methane, which cut the axis of ordinates at the temperature of the flashing point without retardation, and which are asymptotic to the axis of abscissæ at a distance corresponding to the temperature at which slow combustion commences. The part shown in cross-section at the interior of the curve for fire-damp corresponds to the combustion which is terminated at the flashing point, or, in other words, the breaking into flame.⁴

The scientific reason for the retardation of the flashing point should be sought, in my opinion, at least in part, in the fact that methane is an exothermic gas which requires for its dissociation 22.1 calories per molecular weight, and which does not arrive at the flashing point until the moment of dissociation. The necessary calories must, therefore, be supplied by the combustion of a part of the same gas previous to its breaking into flame. In order to give an idea of this we will say that the dissociation of 100 liters of methane requires a quantity of heat equivalent to that produced by the combustion of 32 liters of hydrogen. If this is exact, then a mixture of methane and hydrogen containing 25 per cent. of hydrogen should reach the flashing point without any retardation. In trials which I have made where heat was applied at a single point, the quantity of hydrogen actually necessary was 7.2 per cent. higher.

Be that as it may, it is necessary to consider this phenomena of retardation of the flashing point as a protection which hides a danger and which has often been the cause of serious accidents, because of too great confidence reposed in the resistance to the flashing point of fire-damp. The object of my first researches on this gas was to reach the flashing point of methane by means of incandescent wires. During this study the phenomenon of retardation became manifest.

3. *Flashing of Fire-Damp by Incandescent Wires.*—The flashing of fire-damp by means of metallic wires heated to a red heat by an electric current has been the source of many discussions, but, if one studies this question profoundly, it becomes evident that what appeared at first to be divergencies in reality do not exist as such. In order to study the flashing by means of metallic filaments it is doubtless necessary to employ

⁴ El Gristó en las Minas de carbón, *Premera Conferencia experimental*, 1907, p. 19.

a current of low tension and a circuit without induction, in order that, at the moment of fusion of the metallic wire, sparks shall not be produced by the breaking of the circuit, because these sparks would, of course, produce the flashing of the mixture.

That is why I employed in my experiments a current of 4 volts, obtained from two accumulators with lead bases of the Dinin model, placed close to the metallic wire in order to avoid induction effects from long conductors. Under these conditions it is not possible to produce appreciable sparks upon rupture. They would have occurred if I had employed a resistance in series to absorb the voltage during the dynamic state of the passage of the current, as was often the case in the experiments of my predecessors on this subject. On the contrary, in the experiments described below I have never obtained an explosion by rupture of the wire and subsequent sparking.

But there is a very important cause which has contributed without doubt to the apparently contradictory results of Couriet and Meunier, because these experimenters, according to their own statement, worked constantly on the mixture of 9.5 per cent. of fire-damp.⁵ Now, as the higher limit of the flashing point of fire-damp is 14 per cent. of fire-damp, according to the experiments of Burgess and Wheeler, the mixture used by Couriet and Meunier contained an excess of air of only $14.8 - 9.5 = 5.3$ per cent., which is only 1.05 per cent. of oxygen more than the content required for the higher limit of flashing point. If this 1 per cent. of oxygen should be used up by oxidization of the wire there would be left a non-inflammable mixture. This is the reason why Wullner and Lehmann found the most diluted fire-damp mixtures (6.66 per cent.) the most easily inflammable. It is on these diluted mixtures that I made my experiments by employing at times methane prepared in the pure state by means of carbide of aluminum, or better, natural fire-damp.

Under these conditions I have obtained the following results:⁶

1. With wires of ferro-nickel, 0.3 mm. in diameter, with or without fusion of the wire, I did not obtain inflammation of the most sensitive mixtures of pure methane.

2. With platinum wires, 0.5 mm. in diameter, gradually brought to a red heat, I have obtained inflammation in six consecutive trials with mixtures of from 7 to 7.5 per cent. of natural fire-damp, without seeing the filament melt, although it glowed rapidly at the moment when the explosion took place. With filaments of platinum of 0.2 mm. diameter, I have produced two explosions in three experiments with natural fire-damp.

⁵ H. Couriet and J. Meunier: *Recherches sur l'inflammabilité électrique des mélanges d'air et de grisou*, p. 12, 1906.

⁶ El Grisú en las Minas de carbón. *Primera Conferencia experimental*, 1907, p. 26; *Leçons sur le grisou, Première Conférence expérimentale*, p. 28.

3. With wires of soft iron, of 0.9 mm. diameter, the results obtained are very interesting: In employing a straight filament, either horizontal or inclined, or a filament curved toward the top or toward the base, I have obtained six inflammations in 17 experiments with natural fire-damp of from 7.2 to 7.5 per cent. That is to say, one-third of the experiments resulted in inflammation without fusion of the filament. (In cases where inflammation did not result the filament was, however, heated to fusion.) On the contrary, by employing an iron wire inclined at an angle of nearly 45° , with a spiral tower toward the middle, I obtained five inflammations in six experiments, without fusion of the filament, and in three of these experiments I employed the same filament for three consecutive attempts. In another case I inflamed with a filament of twisted wire a mixture which had not been inflamed by fusion of the straight wire. The explanation which I offer for these facts is as follows:

It will happen in the case of a relatively large filament of iron that its center will be at a higher temperature than the outside, which becomes rapidly covered with a layer of melted oxide of iron. Now, if a point exists where the cohesion is somewhat greater than elsewhere, and where the oxide runs together in the form of a ball or a pearl, the center of the wire will be exposed, and the filament will be partially volatilized and become an oxide in the state of vapor, and will produce a flame which will set off fire-damp. Such an event could happen with a galvanized iron wire at the temperature of the volatilization of zinc, which is 675° —or almost $1,000^\circ$ lower than the temperature of fusion of soft iron. On the other hand, if we expose a slight obstacle to the movement of the fire-damp, for example, by placing a cold filament on the incandescent filament, we would be able to produce the explosion more easily, and we can produce it with certainty if by rolling the wire in the form of a helical we heat the gas from two sides and require it to pass at least twice across the red-hot filament. In this case the explosion occurs at a temperature lower than in the case of a straight filament.⁷

⁷ Couriet and Meunier (*Annales des Mines de Belgique*, 1908, vol. 1, p. 90 to 91) thought that this explanation should be rejected, because they could not admit that, at a temperature lower than that of its fusion, "iron would be volatilized and, oxidizing in a state of vapor, produce a flame," and that, even admitting as possible that the volatilization and the flame took place, it would be as well produced with a straight wire as with a curved wire, and better still with a small than with a large wire. With all respect due to these gentlemen, I submit on this point that, being obsessed with their theory of a gaseous envelope of oxygen, they did not well understand my experiments, which often resulted in a contradiction of that theory. I do not think it necessary at the moment to discuss extensively the theory of Couriet and Meunier, but I would say that it is not possible to admit that fire-damp can be inflamed by radiation alone, because the inflammation must commence at the hottest point, that is, at the contact with the incandescent wire. Now, as a single contact does not suffice, because of its short duration, one should curve the wire in the form of a helical and incline it, to prolong this contact and transform the slow combustion of the fire-damp into a rapid

4. To confirm these experiments I made others with soft-steel wires 0.6 mm. in diameter and with pure methane or pure carbide of hydrogen: In four attempts I did not obtain inflammation with a straight, horizontal filament of about 15 mm. length, but inflamed once with a filament about 25 mm. in length and curved toward the top. With an inclined filament, having three helical turns, I obtained two explosions in two experiments, using the same mixture with which I failed to obtain an

combustion, that is to say, a combustion accompanied by flame, which combustion is, however, only visible at a certain distance from the wire, where there is an unburned part of the mixture. Furthermore, as I have observed previously, the temperature of the curved wire at the moment of the inflammation of the fire-damp is lower than in the case of the straight wire. It is between clear orange and white, instead of welding white or dazzling white. This, therefore, makes the radiation less important and makes entirely improbable the production of a flame by volatilization of iron of the center, which is not exposed as in the case of a straight wire heated to a dazzling white heat. I should remark here that neither the length of the wire employed by me nor the duration of my experiments permitted in any case the absorption by the wire of oxygen from the explosive mixture, so as to render it non-inflammable. The amount of fire-damp in these mixtures was never higher than 7.5 per cent. as I have already stated.

As to the theory of the possible volatilization of the center of large iron wires, which has been suggested by the observation of the easy inflammability of fire-damp by a galvanized iron wire brought only to a dark red heat, I cite the following facts: First, it is without doubt that a large wire, heated by an electric current, will have a higher temperature in the interior than on the surface, because of the loss of heat by radiation and convection. It is because of this fact that Heraeus has substituted in his electric furnace platinum ribbon instead of platinum wire, because the wire becomes broken on account of the high temperature to which its center is submitted, and because this part is most distant from the chamber of the furnace. On the other hand, it is also a fact that because of the great difference in the temperature which exists between the center and the surface of steel ingots in metallurgical works that the molds are stripped in order to roll the steel, without first passing through a heating furnace. To this end the ingots are introduced into soaking pits, which are not heated from an exterior source. It is only necessary to establish an equilibrium of temperature in the mass of the ingots before they can be taken directly to the rolling mill.

In reality, in the case under discussion, the difference of temperature between the center and the surface of a wire is not very great, because oxide of iron melts at $1,350^{\circ}\text{C}$. (Le Chatelier-Boudouard, *Mesuré des Températures Elevées*, 1900, p. 167), and its gradual formation should liberate enough heat to compensate in part for the superficial loss by radiation and so cause the hottest zone to be found near the surface. Now, this layer of oxide, gradually formed, prevents in its turn the rapid oxidization of the wire. If now this layer should suddenly disappear by the accumulation of oxide at one point, the center of the wire would be exposed and, since it is already near its point of fusion, its superficial oxidization would take place in this case very rapidly and the heating would be sufficient to melt it, with partial volatilization. This phenomenon of partial volatilization of iron by its heat of oxidization is comparable to that which takes place toward the end of the second period of the Bessemer process.

I think that these explanations, once given, enable one to admit the possibility of the hypothesis which I have formulated to explain the inflammation of fire-damp by a large, straight, incandescent wire.

inflammation in the three consecutive attempts with the straight, horizontal filament of 15 mm. length. This last experiment was repeated with the same result, employing natural fire-damp.

I think that the results could not be more conclusive, and that, if the flame or the electric spark were the most appropriate means for inflaming fire-damp, it is nevertheless true that incandescent filaments could produce inflammation with equal certitude without the intervention of the flame, on condition that the wires are not melted during the time of the retardation of inflammation of the fire-damp, and provided that the

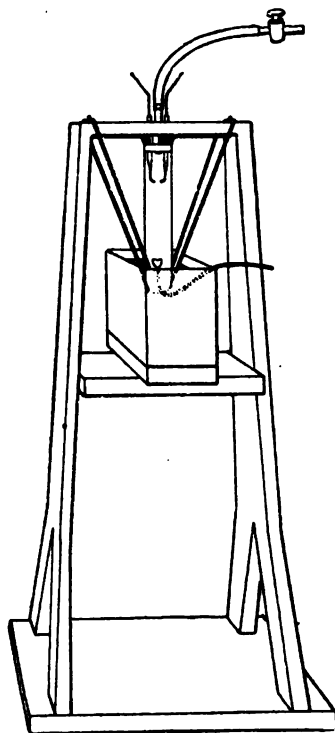


FIG. 2.

temperature of the wire and its heating surface are such that the fire-damp mixture in contact with it can be rapidly brought to the temperature of quick combustion or of inflammation, before it burns slowly or its oxygen is absorbed by the incandescent wire.

The apparatus which I employed for the experiments of inflammation consists, as shown by Fig. 2, of a large cylindrical glass tube of 40 mm. diameter and 155 mm. of useful length,^{*} placed in a bowl filled with water. It is closed at the top by a rubber stopper perforated with three holes.

^{*} Total length, 180 mm.

A glass tube passes through the center hole and is joined to a pet cock for introducing the gaseous mixtures. Glass tubes also pass through the other two holes and terminate in metallic points at their lower ends, between which the spark can be produced. By means of a drop of mercury the connection is established between the metallic points and the wires of the exterior circuit which enter the tubes. By adjusting the height of the tubes and points in the cylinder the spark can be regulated at the desired height, but, to make the experiment more sure, I employed sometimes two pairs of points, one at the upper part of the test tube and the other toward the base. To try inflammation with incandescent filaments, a curved cable with two conductors, duly isolated so that their points are separated in the form of a spark, is introduced at the lower part

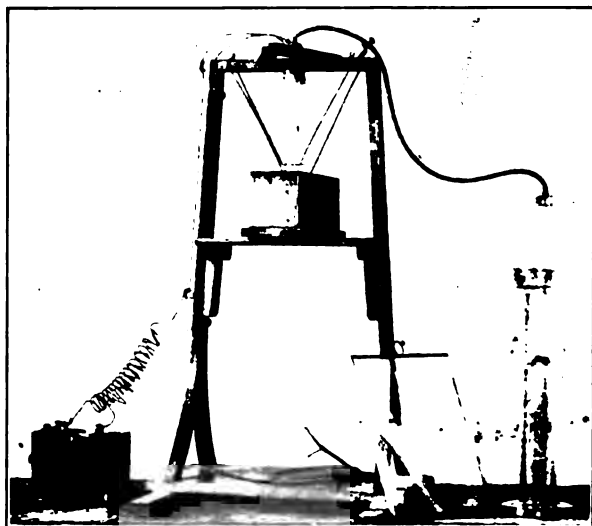


FIG. 3.

of the test tube through the bowl of water. Between the points of these conductors is placed the filament which is to be brought to a red heat by the passage of the electric current. The filament is situated about one-third or one-half of the distance from the bottom of the test tube. The entire system is bound together by iron stays on a solid support of wood, as shown in Figs. 2 and 3.

The results of these experiments have also proved to me that pure methane prepared in the laboratory conducts itself in the same manner as natural fire-damp as to its inflammation by incandescent filaments, from which I have been led to believe that it can be used with an incandescent wire of small diameter and serve as a physico-chemical means of characterizing the influence which other combustible gases have on natural fire-damp.

4. *Influence of Combustible Gases on Properties of Fire-Damp Mixtures.*

—Many discussions have been had on the existence of hydrogen and hydrocarbons in fire-damp, as indicated by chemical analysis. The discussions have involved especially the consequences which the presence of these gases, more inflammable than methane, would have upon our knowledge of the properties of so-called "sharp gas." For my part, I believe that, instead of entering into academic discussions based on samples of gas which it is impossible to procure twice under the same conditions, it would be much better to study the properties of synthetic fire-damp consisting only of pure methane, and then to examine the influence exercised upon the properties of this gas by the addition of variable quantities of the combustibles mentioned above, in order to be able to determine at just what point their mixture with fire-damp would augment the danger.

This, however, has not prevented me from studying the question of the analyses of fire-damp, of which I will speak later.

The inflammable gases which could accompany methane in fire-damp, through circumstances which are fatuitous or a little understood, are ethane, ethylene, and hydrogen.* None of these gases have an appreciable retardation of the flashing point, so that it would be easy for us to study their influence on fire-damp by preparing mixtures of each of them with methane in different proportions and seeking to inflame them with metallic wires of a diameter which would be incapable of inflaming methane alone.

From the conclusion that the most oxygenated mixtures are the most easily inflamed, I have always attempted in my experiments to maintain mixtures close to the lower limit of inflammability, without however always accomplishing it. I have obtained the following results, which I consider as simply a step in advance, but which give an idea of the influence exerted by the presence of other gases on fire-damp mixtures.

Of ethane, whose limit of inflammability was 3.9, I employed 4.5 parts with 1.82 parts of methane, in order to accomplish inflammation by the fusion of a ferro-nickel wire of 0.3 mm. diameter. In this case the volume of ethane was 66 per cent. of the total volume of the two inflammable gases.

Of ethylene, whose limit of inflammability was 3.6, I used 4 parts with 2.85 parts of methane to obtain inflammation under the same circumstances. In this case the ethylene was 58.5 per cent. of the total volume of the two inflammable gases.

Of illuminating gas, with a limit of inflammability of 8.5, I obtained inflammation using 5 parts mixed with 4.5 parts of fire-damp, the illu-

* El Grisú en las minas de carbón. *Primera Conferencia experimental*, 1907, p. 44.

minating gas being, therefore, 54 per cent. of the volume of the two inflammable gases.

With hydrogen and methane I obtained inflammation with a mixture containing 2.9 parts hydrogen and 6.1 parts fire-damp, the hydrogen therefore representing 32.2 per cent. of the total volume of the inflammable gases.

Thus we see that a large quantity of the foreign gas is necessary to cause methane to lose its retardation of flashing point under the conditions existing in my experiments. This does not prevent a relatively small quantity of these same gases from extending the limits of inflammability of fire-damp mixtures, but it is interesting to note that methane conducts itself in this case like a true paraffin in diminishing the sensitiveness of the inflammable mixtures, as is the case with gun cotton impregnated with paraffin, which requires a much stronger detonator to detonate it than if it were in the dry state.

The relatively large quantity of these gases, which it is necessary to add to pure methane to render it inflammable by means of a thin metallic incandescent wire, proves that the presence of traces of hydrogen or of ethane reported by chemical analysis would not have any appreciable influence on the characteristic properties of fire-damp. In any event, this natural gas should not be protected by a "sacred taboo," which would take away the value from experiments made with methane sufficiently pure, and which can be prepared at a low net cost by a method which I will indicate later. On the contrary, the possible influence on the properties of natural fire-damp in fuel gases of hydrogen and oxide of carbon should be taken into account.

5. *Influence of Inert Gases on the Properties of Fire-Damp Mixtures.*—The influence of nitrogen and carbonic acid on the limits of inflammability of methane is only appreciable as the diminution in the proportion of oxygen is felt. My experiments, in accord with those of Le Chatelier, show that each 1 per cent. of carbonic acid added to the air up to 8 per cent. raises the lower limit of inflammability by about $\frac{1}{1000}$. But the case is a little more complex in mines where the gas coming from the operation of gobbing produces an enrichment of the air in the mines in carbon dioxide and water vapor at the expense of the oxygen of the air. The upper limit of inflammability descends rapidly and the lower limit is raised slowly in direct ratio as the content in oxygen is diminished, until we have only a single value toward 13.5 per cent. of oxygen with a content of 6.7 per cent. of methane. Without going further in this discussion I should say that I have abandoned the studies made by me in this way after having

¹⁰It is appropriate to compare here the resemblance of these results with illuminating gas to those reported by the French Commission on Fire-damp in its experiments on the inflammability of fire-damp by sparks obtained from a blow of steel on steel.

noted the exhaustive work done in the United States by J. K. Clement.¹¹

But there is still another interesting point to note; namely, the influence of carbon dioxide on the diminution of the rapidity of combustion of fire-damp mixtures, of which it forms a part and from which one can draw material for useful deductions. I will recall first that, as noted by F. Clowes,¹² the composition of air when it leaves the flame of a candle or of an oil or alcohol lamp is comparable to that exhaled from the lungs. Now, since the gas from gob areas is also the result of a slow combustion, it is easy to see that the air exhaled from the lungs could be substituted in the laboratory¹³ in experiments with fire-damp, of which those which follow have a special interest:

Let us now consider the relighting, without danger, of safety lamps in a fire-damp atmosphere.¹⁴ The relighting of safety lamps in a mine has been the cause of many discussions and Marsant¹⁵ has said relative to materials for relighting by friction, which are considered the safest today (as based on the experiments at Frameries on this subject):

"But for those (relighters of white phosphorus) the experiments at Frameries and elsewhere do not give the certainty that the lamps submitted to the proof were completely freed from inert gas before the relighting. At Frameries prompt relighting with wire gauze made red hot or very warm especially occupied attention, but under these conditions could one be sure that the lamp is free from all inert gases of former combustion at the moment of relighting? The very capricious results of the tests allow one to doubt, or even to think the contrary.

"The non-explosive phosphorus relighters which are supposed to be almost inoffensive, were not submitted at Frameries to a sufficient number of tests to lead to a sure conclusion."

In consequence of the above I am led to believe that, if we could assure ourselves that the atmosphere of the lamp at the moment of relighting was under the same conditions as when we studied this relighting, the problem would be solved. Now, the gas exhaled from the lungs mixed with fire-damp diminishes considerably the latter's rapidity of combustion, which becomes comparable to that of mixtures near the limit of inflammability. Then, if in a safety lamp which has been extinguished some time and which no longer contains inert gases of combustion we inject air from the lungs, one would then be able to fill the lamp with a fire-

¹¹ *Technical Paper No. 43 of the U. S. Bureau of Mines.*

¹² *The Detection and Measurement of Inflammable Gas and Vapors in the Air*, p. 168.

¹³ Air exhaled from the lungs has the following composition:

CO ₂ = 4.4 per cent.	Equiv. alent to	CO ₂ = 4.4 per cent.	} = 27.3 per cent. (black damp)
N = 80.4 per cent.		N = 22.9 per cent.	
		57.5 per cent.	
O = 15.2 per cent.		O = 15.2 per cent.	} = 72.7 per cent. (air)

(*Primera Conferencia experimental*, 1907).

¹⁴ *El Grisú en las Minas de carbón*, 2^a *Conferencia experimental*, 1908, p. 125.

¹⁵ *Bulletin de la Société de l'Industrie Minérale*, Mar. 21, 1907.

damp mixture which is inflammable, but only feebly explosive, and non-capable of propagating the flame from the exterior.

After that it would be easy to judge the effect of the exhaled air on the combustion of a safety lamp by blowing softly on the top, an operation which could be executed with very little practice. From another point of view I have been able to observe with an acetylene safety lamp different phases of the extinction of fire-damp mixture through its impoverishment in oxygen and its mixture with residual gases of its own combustion, by working as follows: To make this experiment, which has never been published before, I used a lamp of the Tombelaine system with low admission, which permits the flame to be reduced at will, to examine the aureoles.¹⁶ If now, having reduced the flame to merely a luminous point, we introduce the lamp under a glass bell of about 15 to 20 liters capacity, half filled with methane or fire-damp, and gradually raise the lamp, we see the aureole first elongate and the inflammation *en masse* of the gas contained within the gauze. But instead of seeing the flame extinguished, as would be the case of an acetylene flame thrust into an atmosphere which contained 12 per cent. of oxygen,¹⁷ we see the gas first inflame and then become extinguished at the wick; then as the air entering at the bottom of the lamp dilutes the products of combustion, we see the acetylene flame form a new aureole which is elongated in its turn with inflammation and new extinction of the fire-damp mixture in the same manner as before; and thus we could repeat the same thing several times, if desired.

These experiments allow an answer to be made to a question often asked by mine foremen, namely, why one sometimes sees fire-damp suddenly inflame at the interior of lamps, accompanied by an explosion, while at other times there is formed only a very much elongated aureole without sudden extinction. The answer is easy: If the air of the mine is pure its mixture with fire-damp is at first explosive, but, if the air of the mine is rich in black damp, as the residual from respiration or from its mixture with gas from gobbing, the aureoles are more or less elongated and the inflammations in the interior of the lamp do not become true explosions.

I think that an extension of these considerations would give us an explanation of "sharp gas."

Preparation of Pure Methane by Means of Commercial Carbide of Aluminum

Admitting that the properties of fire-damp as an inflammable gas could be studied starting with pure methane, it remains to explain the

¹⁶ Tombelaine: *Informe sobre la lámpara minera de seguridad de Acetileno sistema*, 1912.

¹⁷ 11.9 of O_2 + $7CO_2$ + $81.1 N_2$ (according to Beyling). It is interesting to note in this connection that methane will not burn when the mixture contains less than 14 per cent. of combustibles.

method of preparing this gas under conditions of purity necessary to this end and at a price sufficiently reasonable for the purpose.

My predecessors in these studies have not made sufficiently clear the fact that methane prepared by means of the acetate often contains unsaturated hydrogen and hydrocarbons,¹⁸ and that prepared by means of the commercial carbide of aluminum may contain considerable quantities of hydrogen, especially if it is produced hot.¹⁹ It remains to find other means to purify the gas, which would require a second operation, or to employ pure carbide of aluminum prepared by the Moissan process, costing 400 francs per kilogram. This would make a cubic meter of pure methane at 0°C. and 760° pressure cost about 1.430 francs. For these reasons I have undertaken another study in which I have succeeded in preparing pure methane by means of commercial carbide of aluminum which only costs 3.30 francs per kilogram.²⁰

The principal impurities of carbide of aluminum are the carbides of calcium, iron and silicon, together with an excess of aluminum and uncombined carbon and of alkalies with a little sulphide of aluminum, and perhaps silicide of aluminum.

The carbides of aluminum and of silicon are not attacked by cold water so that we are not handicapped by their presence. As carbide of aluminum is only attacked slowly by cold water the problem resolves itself into the separation of the carbides of calcium and the alkalies by a simple treatment with cold water, provided the hydrates of lime and alkali liberated by this reaction, and which remain inclosed in granules of carbide of aluminum, do not later act on the metallic aluminum in excess, which the carbide contains and produce hydrogen, because it is this gas mixed with methane which modifies most strongly the properties of the latter gas. But even in the absence of alkalies, aluminum decomposes water slowly by virtue of the following thermic equation:



This reaction is more easily disturbed if the aluminum contains traces of calcium or alkali metals. On the other hand, the reaction once commenced will be immediately lessened in rapidity on one side by a slight layer of hydrate of aluminum which covers the metal, and especially by the presence of little bubbles of hydrogen which cover it. These are easily disengaged either by heating or by forming a vacuum above the liquid. By taking note of the different facts which I have discovered, I have deduced the following method for the purification of carbides:

¹⁸ See *Leçons sur la Grisou*, *Première Conférence expérimentale*, pp. 13 and 14.

¹⁹ El Cristó en las Minas de carbón, *Segunda Conferencia experimental*, p. 9, *et. seq.*

²⁰ For more details of this process of purification see *Segunda Conferencia experimental*, pp. 7 to 21, and a note published in *Anales de la Sociedad Española de Física y Química*, May 5, 1913.

It is necessary first to take out the excess of aluminum and this can easily be done by pulverizing the carbide between the cylinders of a rolling mill which, at the same time, flattens out the metallic aluminum. It then only remains to sift the mixture to separate the good part from the metal.

To get rid of the carbide of calcium, the sifted carbide of aluminum is treated several times with water until the supernatant water is clear. As the hydrate of aluminum is only slightly soluble in water we transform it into a very soluble chloride by means of washing with dilute hydrochloric acid. For this purpose use 5 per cent. of acid of 1.19 specific gravity, then continue washing with water until the wash water no longer is acid. For 50 grams of carbide we would require about three washes with 500 c.c. of water before the treatment with acid and as much again afterward.

After this purification the carbide is in condition to be used, but one should also add the precaution of rapidly drying by carrying off the water by means of alcohol followed by one washing with ether.

In order to obtain from this carbide a methane almost free from hydrogen it is necessary to avoid as much as possible the disengaging of little bubbles of hydrogen which cover the aluminum during the attack on the carbide. This can be done by giving a certain thickness to the mass of carbide on the bottom of a conical flask where the reaction is produced. If, on the other hand, we give a certain height to the water which the gas has traversed to reach the gasometer one will then find it in its best condition and one can obtain in winter, with the temperature about 14°C., a methane almost free from hydrogen (about 0.2 per cent.).

But this process becomes incomplete in summer where sometimes I have found in the methane as much as 8 per cent. of free hydrogen. It is easy to see that under these conditions the liquid has become alkaline, its analysis showing it to contain aluminate of soda.²¹

To prevent this occurrence it is sufficient to replace daily the water which covers the carbide by washing until the wash waters are no longer alkaline. Operating thus we can obtain even in summer methane containing less than six parts per thousand of hydrogen, which is available for the greater part of the experiments. For the work of analysis it suffices to purify the gas by washing it with potash and passing it through a tube containing palladium.

²¹ For further details on this subject see my "Nota sobre la obtención del metano puro mediante el carburo de aluminio comercial." *Anales de la Sociedad Española de Física y Química*, May 5, 1913.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 59th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Magmatic Differentiation in Effusive Rocks

BY SIDNEY POWERS, PH. D., CAMBRIDGE, MASS., AND ALFRED C. LANE, SC. D., TUFTS COLLEGE, MASS.

(New York Meeting, February, 1916)

INTRODUCTION

THIS paper aims to present the results of an investigation concerning gravitative differentiation in lava flows, based on a quantitative microscopic and chemical study of a Triassic basalt from Nova Scotia, and confirmed by data secured from a study of certain Keweenaw ophites.

At Cape d'Or, Nova Scotia, near the head of the Bay of Fundy, marine erosion has exposed a series of Triassic basalt flows in which copper has been sought for a number of years. Studies of the thickest flow of the series with regard to volumetric composition, specific gravity, and grain have been made by the junior author, and measurements by students at Tufts College, together with confirmatory observations of the same nature on this and other flows by the senior writer, and chemical analyses made by the Division of Mines, Mines Branch, Ottawa,¹ all show differentiation in lava flows comparable to that in intrusive bodies. The work was made possible by the availability of the drill-cores, the property of Tufts College, and emphasizes the desirability of the study of drill-cores from other localities. Measurements of the specific gravities of certain Keweenaw lava flows of Michigan by the senior author, from material collected in 1915, will be given as a comparison to those of the Cape d'Or flow.

It has been found that the specific gravity of drill-cores, or volume per cubic foot, can be readily obtained by measuring their dimensions. This can be done to an accuracy of within 1 per cent. in pieces of core over 100 mm. in length by using a micrometer gage for the diameter, and taking the average of six measurements of the diameter; and the average of four measurements of the length made with a finely graduated ruler. Then, after weighing them, the weight in grams per cubic centimeter, or what is practically the same thing, ounces per cubic foot, is obtained. System-

¹ This paper is an abstract from a report on the copper at Cape d'Or by Alfred C. Lane with notes on the volume composition of the rocks by S. Powers, to be published in a "Report on Copper Mines and Mining in Canada," by Dr. A. W. G. Wilson of the Department of Mines; and the chemical analyses, by M. F. Connor, are reproduced by permission of Dr. Wilson and of Dr. Eugene Haanel, Director of Mines.

atic tests seem to show that this may be of considerable practical value, as the variation in one lava flow between the denser, more crystalline, less glassy center and the more glassy, less crystalline, and perhaps more altered upper and lower parts, can be distinctly and continuously followed when the amygdaloidal or vesicular structure is not conspicuous.

Differentiation in effusives has previously been studied by the senior writer² and by him and Queneau³ the relation of size of grain to the position in the flow. The former described gravitative differentiation by fractional crystallization and gravitative separation of the feldspar to the top and of the olivine and pyroxene to the bottom in Keweenaw flows of Michigan.

Differentiation in corresponding intrusives has been described by Lewis⁴ in the Palisade diabase, where the top and bottom of the sill were quickly chilled and therefore have an average composition, while differentiation took place in the remainder of the magma with the formation of an olivine-rich layer above the quickly cooled base, and a concentration of the augite below the middle of the sill, and of feldspar above the middle.

Cape d'Or Flows

Cape d'Or is a conspicuous headland projecting into Minas Channel at the entrance to Minas Basin, and rising to a height of 200 ft. above mean tide level. The sea cliff is composed of massive basalt, columnar near the shore, and is being cut back by wave attack and by tidal currents which sweep back and forth, because the range of tides is about 37 ft. at the Cape, and much more inside Minas Basin. Above the cliffs is a rolling plain upon which stands the plant of Colonial Copper Co. (now idle). On the north is an escarpment formed by the erosion of the base of the lava flows and of the underlying Triassic sandstones and shales, both of which dip south at a low angle. The escarpment faces a low plain, underlain by Triassic sandstone and Pennsylvanian sediments, that is separated from the Cobequid Mountains by a fault-line scarp. Erosion and normal faults have separated Cape d'Or from the remainder of the Triassic, so that it now has the appearance of a horst.⁵

During the course of mining operations, two shafts were sunk into the basalt, trenches were dug across the basal contact of the flows with the

² A. C. Lane: *Bulletin of the Geological Society of America*, vol. viii, p. 403 (1897); vol. x, p. 16 (1899); vol. xiv, pp. 369 to 384 (1903); *Michigan Geological Survey*, vol. vi, Part I, p. 106 (1898); *Michigan Geological and Biological Survey*, Pub. 6, vol. i, p. 145 (1909). See also *Trans.*, vol. xxxviii, p. 931 (1907).

³ A. L. Queneau: *School of Mines Quarterly*, vol. xxiii, pp. 181 to 195 (1902); *American Journal of Science*, 4th Ser., vol. xiv, pp. 393 to 396 (1902).

⁴ J. V. Lewis, Petrography of the Newark Igneous Rocks of New Jersey, *Annual Report for 1907*, *New Jersey Geological Survey*, p. 132.

⁵ S. Powers: The Acadian Triassic, *Journal of Geology*, vol. xxiv, 1916.

underlying sediments on the north, and seven holes drilled with diamond drills at the localities shown on the map (Fig. 1). The basalt was found to comprise five flows, with a total thickness of about 770 ft. Of this series the lowest flow is the thickest, 556 ft., and the measurements of volume, specific gravity, and grain, together with five of the chemical analyses were made from the drill-cores of it. The extrusion of this thick flow was followed in a relatively short period of time by the pouring out of three thin flows, ranging in thickness from 11 to 75 ft. With a new spurt of volcanic activity another thick flow appeared, the amygdaloidal top of which has been eroded away, leaving a thickness of only 135 ft.

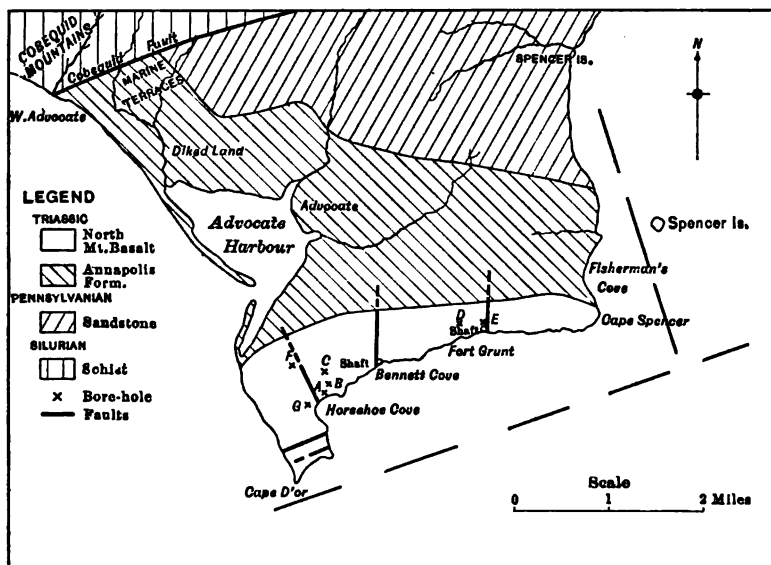


FIG. 1.—MAP OF THE CAPE D'OR REGION, NOVA SCOTIA, SHOWING THE LOCATION OF THE DRILL-HOLES A TO G.

Measurements were made on core A, from near Horseshoe Cove, and on core F, half a mile to the north. (Reproduced by permission from the *Journal of Geology*, vol. xxiv, 1916.)

Some of these flows are separated from one another by a slight amount of other material, very difficult to find on account of its paucity, which may have originally been tuff or dust as the flows spread over a land surface.

The separate flows are sometimes distinguished with difficulty both in the field, in the Triassic area in general, and in the drill-cores. Each flow is characterized by an amygdaloidal top, and a thinner amygdaloidal base. The center may or may not be vesicular: in the thicker flows it is always dense, and yet it may have vesicular streaks (as in one of the flows near Meriden, Conn.), which represent the places where the rising bubbles of gas were caught against the cooling upper portion of the flow. Usually these vesicular streaks can be distinguished from the vesicular

or amygdaloidal top of the flow by the lack of any break and the coarse grain; but when they appear near the top, where the whole rock is vesiculated, it is very difficult to find the true top in a more or less fragmentary drill-core. Thus, in hole *F*, described below, the boundary between the two lower flows cannot be identified with certainty, and a thin flow, not represented in the other cores, may appear here. A further criterion for the top of a flow is the relative amount of alteration—greater than elsewhere—giving a lower specific gravity as shown in Fig. 4. A coarseness of grain characterizes the center of thick flows (Fig. 3), but even where this is not noticeable in the field, as in the lower flow of the First Watchung basalt near North Plainfield, N. J., a bed of tuff may separate the flows, as at that locality.

The thickness of the different flows at Cape d'Or in feet, as shown in the drill-cores, follow. Hole *D* passed through a fault-breccia and is therefore omitted. The location of the drill-holes is shown in Fig. 1.

Flow	Character	Drill-core					
	(Top Eroded)	A	B	C	E	F	G
Fort Grunt.....	Massive	103	135	132	132	81	135
		103	135	132	132	81	135
Fourth.....	Amygd.	6					
	Massive	6					
		12 115	11 146	12 144	12 144	11 92	18 153
Third.....	Amygd.	5	5			2	3
	Massive	39	39			9	38
		44 159	44 190	46 190		11 103	41 194
Goose Tongue roof....	Amygd.	20	20	18	21	13	21
	Massive	35	46	29	35	18?	44
		55 214	66 256	47 237	56 200	31? 134	65 259
Cape Spencer.....	Amygd.	20	20	19	20	26?	16
	Massive	536	(136) +	(104) +	(51) +	(520) +	(205) +
		556					
Total thickness.....		770	412 +	360 +	271 +	680 +	480 +

The Fort Grunt flow is composed of massive basalt, except for the lower few feet, which shows the effects of alteration throughout, and of weathering at the upper, erosion surface. It is characterized by porphyritic labradorite (Ab_2An_3) crystals 1 to 3 mm. long; augite prisms 1 to 2 mm. long and often 0.5 mm. thick; magnetite, ilmenite, and hematite; enstatite and olivine only near the base (44 and 4 ft. from the base); and glass. The principal alteration minerals are hematite, limonite, chlorite,

and malachite. The chemical composition is given by analyses C-13, 77, 105, 128. The corresponding thin sections show: in C-13, relatively abundant glass, alteration with the development of chlorite and limonite, although the analysis shows a high ferrous iron content; in C-77 a fresh rock with little glass; in C-105 slight alteration with more glass; and C-128, only 4 ft. from the base, a fine-grained rock with more alteration.

The Cape Spencer flow is more massive than the Fort Grunt flow, with well-developed columnar jointing. The basalt is similar to that of the latter flow, except that glass is almost lacking in the center of the flow, alteration is not so great, and the grain is coarser. The chemical analyses are C-325 (88 ft. down), A-387 (167 ft. below the top), A-492 (272 ft.), A-597 (377 ft.), A-706 (486 ft.). C-325 is a fresh basalt with abundant glass, a fine grain and some olivine; A-387 is less glassy, with more feldspar than augite, and no olivine; A-492 and 597 exhibit a comparatively coarse grain, an excess of augite over feldspar, an absence of glass and olivine; A-706 shows what is apparently a trace of micropegmatite. At A-450 (230 ft.) an inclusion of fine-grained basalt was found in the moderately coarse basalt. At A-553 (333 ft.) there are several generations of crystals, and the finer intergrowths, the last to crystallize, are probably composed of plagioclase and orthoclase with minute crystals of apatite. About 5 per cent. of the rock is composed of olivine of the same generation as the normal augite and feldspar.

The Cape Spencer flow rests on sandstone and shale of white color near the flow, but becoming red with only white streaks 10 ft. below the base. At the very base are a number of magnetite octahedra. A bed of rather soft bluish-green material also occurs at this place; it is chlorite, which may have replaced a zeolite.

Copper occurs in the flows in small quantities throughout the basalts as the analyses show, and in some of the amygdules, but principally in the veins or belts of fault breccias, especially from near the base of the Fort Grunt flow to a short distance down in the Cape Spencer flow, where it was deposited by circulating waters as native copper and as malachite.

Volumetric Composition

A series of volumetric Rosiwal measurements was made upon the Cape Spencer basalt in core A in order to compare the results with the chemical analyses. In Fig. 2 the results are plotted and curves are drawn to indicate the variations with depth in the quantity of feldspar, augite, and glass, with the magnetite and other iron ores included with the glass. In spite of the small number of thin sections measured, it is apparent that there is a concentration of feldspar near the top of the flow; a slightly greater percentage of augite in the center, and of feldspar near the base; a large amount of glass at both the top and bottom, but more

at the top; a rather uniform quantity of iron ores throughout; and the presence of olivine only at the top (except the abnormal rock at 333 ft., described above, which may well be a sunken portion of the crust). It is probable that the more quickly chilled top and bottom of the flow, with almost equal amounts of feldspar and augite, represent the original composition of the magma.

In making the necessary measurements, difficulty was found by both authors in distinguishing the feldspars from a glassy rim which frequently

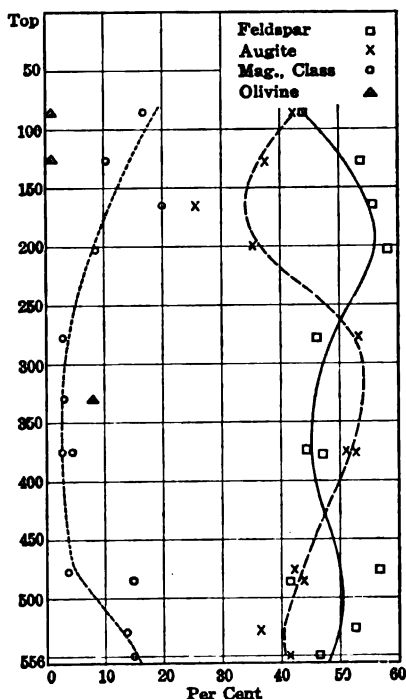


FIG. 2.—CURVES SHOWING THE VARIATIONS WITH DEPTH IN THE VOLUMETRIC COMPOSITION OF THE CAPE SPENCER BASALT, CAPE D'OR.

Feldspar is represented by the solid line, augite by the dashed line, magnetite and glass together by the dotted line.

surrounds them, especially near the base of the flow. It is best, therefore, that the curves be understood to represent tendencies, and that their qualitative value is greater than the quantitative.

The significance of the rising of the feldspar and of the settling of the augite, together with the appearance of the olivine only at the top are discussed elsewhere.

Variation in Grain

Measurements of the grain of rocks has previously been attempted by the senior author in the case of Keweenaw basaltic rocks, and by

Queneau in the Palisade diabase, as already referred to, and the theory of grain has been discussed in detail by them.

There are several ways in which to measure the grain, and several ways in which to plot the results. The method which has been used here is to measure both the length and breadth of the largest crystals of both augite and feldspar in each slide and plot the results in areas. It seems necessary to measure both dimensions in view of the fact that certain crystals grow in different directions in different stages of develop-

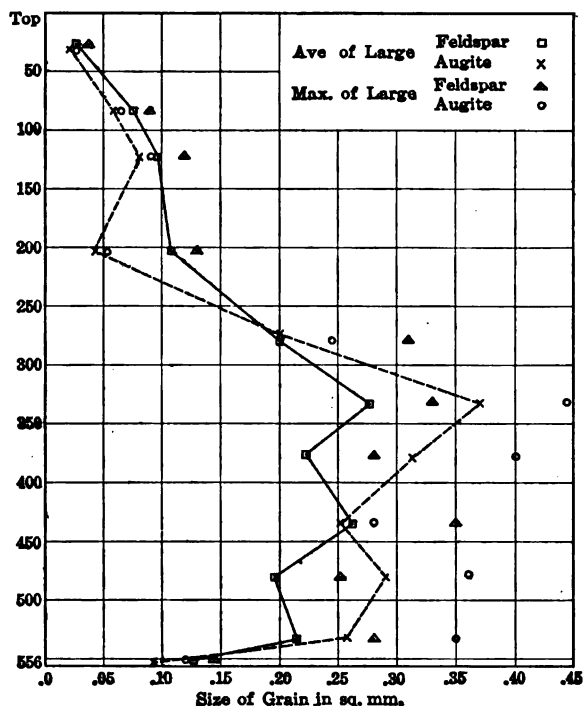


FIG. 3.—THE VARIATIONS WITH DEPTH IN THE GRAIN OF FELDSPAR AND AUGITE IN THE CAPE SPENCER BASALT.

The average area of the five largest crystals of each kind are plotted and the points connected by a continuous line in the case of the feldspars, by a dashed line in the case of the augite. The area of the maximum crystal of each kind for each slide is also plotted.

ment. Another method is to make Rosiwal measurements of the entire slide, and divide the total of the readings for a given mineral by the number of readings. This method is, however, open to the objection that several generations of crystals are usually represented in amounts varying irregularly in each slide, and that glass is not readily distinguished from some of the crystals, nor is one feldspar from another under certain conditions, especially in thin slides. The results may be plotted in terms of an average linear unit, as was done by the senior author in the Keweenaw ophiolites, instead of in areas, as is done here.

In order to secure the greatest degree of accuracy, from five to fifteen of the largest crystals of both minerals in each slide were measured in order to show the existence of phenocrysts, which must not be included in the results because they represent an earlier period of crystallization. The difficulty is to select the phenocrysts, and upon the personal judgment the accuracy of this method must depend. In order to check the largest normal crystals, the average of the largest five are plotted in Fig. 3, and also the maximum (excluding phenocrysts). In actual practice it is found almost impossible to obtain the same results, quantitatively, on two measurements of the same slide. Hence, the degree of accuracy of the method is slight, and the results show merely a tendency to develop a certain grain and do not show the absolute grain. This point is further emphasized by the irregularities in the points as plotted, for curves connecting these points would cross and recross in three successive measurements.

The points show a fine grain at the top of the flow, and a coarse grain at or near the middle, with apparently a sudden transition between the two. At the base of the flow it is almost impossible to tell what is the true grain, on account of the phenocrysts, but there is apparently another sudden change to a finer grain.

Both the determinations of differentiation by volumetric measurements and by variations in grain are rather unsatisfactory, because there appears to be such a wide variation of results in any set of measurements. The methods employed are not as much the cause of this variation as is the material. The lava flow is subject to rapid chilling, and the processes of both differentiation and growth of grain are interrupted by this chilling. Also, there appear to be streaks which are either not affected by the differentiation, or else have an abnormal grain, even when they are in the center of the flow. These irregularities may be readily referred to differences in temperature and composition produced by flow currents, by engulfment of parts of the crust, and by cracks and cooling from them while the rock was yet hot and not completely crystallized. In the case of thick sills, more satisfactory results should be obtained by using the same methods of measurement.

Variation in Specific Gravity

In order to determine the relation of the specific gravity of the basalt in the Cape Spencer flow to the depth and to compare the results with those for the volume composition and grain, the specific gravities, or volumes in grams per cubic centimeter, which is the same if porosity is neglected, of 36 specimens from core *A* and 20 from core *F* were measured, the latter by students at Tufts College under the direction of the senior author. In core *A*, five of the determinations were made with a Joly

balance; eight by the usual method of immersion with a two-arm balance; and the remainder by a method of measurement (given above on p. 1) which had previously been used by the senior author. The first and last methods were checked by the other using the same specimens in some instances and were found to be sufficiently accurate considering the nature of the material.

The specific gravity is affected by the porosity, but the porosity of basalts and diabbases where free from amygdules and vesicles is low.*

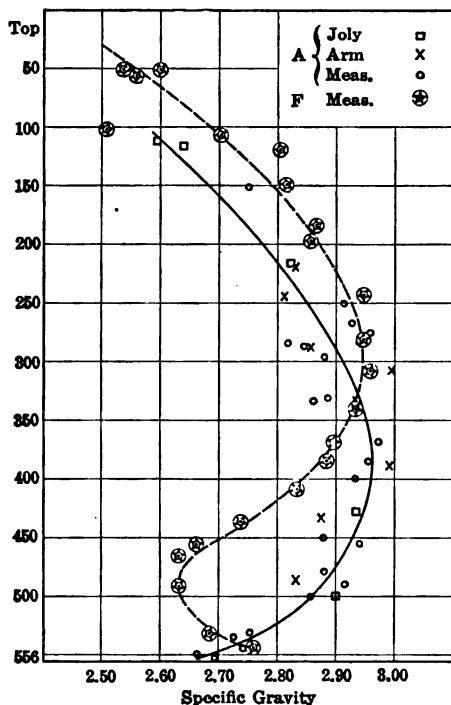


FIG. 4.—CURVES SHOWING THE VARIATION WITH DEPTH IN THE SPECIFIC GRAVITY OF THE CAPE SPENCER BASALT.

The data was secured as indicated by the use of a Joly balance, a two-arm balance, and a system of measurements. The curve for core A is a solid line, that for core F a dashed line.

If the porosity is neglected as a uniform quantity in the dense basalts, the other factors which must be guarded against are: The vesicles and amygdules; zeolitization; chloritization; and other forms of alteration, either in the rock or along veins or joints.

Curves showing the averages of the measurements of core A and of core F are shown in Fig. 4 by the solid and dashed lines, respectively.

* The porosity has been measured by the senior author (*Annual Report for 1909, Geological Survey of Michigan*, vol. i, p. 100).

Both curves show a marked increase in specific gravity at or just below the center of the flow with a gradual decrease above and a more sudden decrease below this point. The discrepancies between the two curves are: The higher position in the curve for *F*, and the slight increase in specific gravity at the base of the curve for *F*. The curve for *F* is plotted on the assumption that the drill passed through 546 ft. of this flow. As hole *F* was bored half a mile from hole *A* and the underlying sediments were not cut, this is quite likely not exactly comparable. The irregularity at the base may also be due to local conditions at this spot in the flow. The points determined for core *A* show a great variation near the center where the basalt is entirely free from zeolitization and alteration and where long cores were available. This must be accounted for partly by streaks of heavier and lighter rock, partly by inaccuracies in measurement. At the base of the flow, chloritization and other forms of alteration give a lighter rock; at the top, zeolitization and alteration combine to give lower figures.

In order to compare the range of specific gravities along the curves, from 2.60 to 2.96, with that in other basalts the following figures are given: Kilauean basalt 2.75; tachylite 2.58; Watchung basalt 2.91 to 2.99; Mt. Holyoke basalt 2.97; and the Keweenaw figures given below.

A comparison with the results for volume composition and grain shows a marked agreement between the maximum concentration of the augite, the maximum grain and the maximum specific gravity—all slightly below the center of the flow.

The Keweenaw basaltic effusives have precisely the same differentiation, as already pointed out by the senior author, so that the hanging-wall trap of the amygdaloid lodes is easily distinguished from the foot-wall trap. Here again the upper part of the flow contains more feldspar and the lower part more augite. The upper part of the flow is then more feldspathic and tends to a porphyritic (seriate porphyritic or glomeroporphyritic) texture. This is the foot-wall trap. The lower part of the flow that contains more augite tends to a luster-mottled poikilitic ophitic texture. This is the hanging-wall trap. Most of the flows if thick enough show a trace of ophitic texture at some depth. In his 1899 paper the senior author gave analyses, not here repeated and the specific gravity of the lower ophitic part was given as 2.877, of the upper part as 2.781. Later measurements of the weights of the drill-cores from Keweenaw Point in grams per cubic centimeter (*i.e.*, tons per cubic meter, thousands of ounces per cubic foot) show that these figures can be used as an index of differentiation precisely as in the Cape d'Or flows. Ophites give figures from 2.87 to 2.96. Feldspathic melaphyres, representing the upper feldspathic part, for instance, the foot walls of the lodes of the Quincy mine, are less heavy (2.75 to 2.78). Values below 2.75 in a rock not obviously porous generally indicate more of a felsite,

or zeolitization. The mean value of the specific gravity of trap was found by McNair to be 2.8865 varying from 3.0904 to 2.7623. The mean value of 50 samples of amygdaloid was 2.8454 varying from 3.0936 to 2.7034.

E. Koepel, superintendent of the Champion mine, found the specific gravity of the rock tailings from that mine—running but 0.10 to 0.06 per cent. copper and with little quartz, calcite, or epidotes—to be 2.84.

Chemical Composition

The chemical analyses of the Fort Grunt and of the Cape Spencer flows show closely related rocks (Bandose to Beerbachose), and the differences between the analyses of various specimens from a single flow are greater than those between the average of the two flows. The magnesia is generally lower in the Fort Grunt flow as though this flow might have been an effusion from the same source after more of the olivine had settled out.

In the Fort Grunt flow the first and last analyses represent slightly altered rocks. Below C-13 the alumina and ferric iron increase; the silica, ferrous iron, magnesia, and lime decrease. In the Cape Spencer flow there is an increase with depth from A-387 down in ferrous iron, lime, and magnesia; the concomitant changes in potash and in silica are within the limits of error, but there is a decrease in ferric iron. As the Fort Grunt analyses represent only the lower portion of the original flow, they should be compared with the base of the Cape Spencer flow. In the former there is more ferric iron, potash, and titanium, less ferrous iron, magnesia, and lime, *i.e.*, less pyroxene.

	Fort Grunt Flow				Cape Spencer Flow		
	(1) C-13	(2) C-77	(3) C-105	(4) C-128	(5) C-325	(6) A-387	(7) A-492
Distance from base of flow, ft.	119.0	55.0	27.0	4.0	437.0	389.0	284.0
SiO ₂	50.14	54.04	52.98	52.30	52.50	53.00	51.92
Al ₂ O ₃	14.56	13.82	14.08	14.86	14.30	14.71	13.25
Fe ₂ O ₃	4.40	3.25	4.02	5.38	6.15	7.30	2.28
FeO.....	7.28	6.95	6.34	5.92	5.14	3.32	7.16
MgO.....	5.46	6.34	5.44	5.55	6.08	4.67	9.05
CaO.....	10.10	10.12	9.72	9.80	9.08	9.40	11.22
Na ₂ O.....	2.59	2.51	2.93	2.17	2.56	4.11	2.42
K ₂ O.....	0.55	1.20	0.96	1.36	0.66	0.71	0.64
H ₂ O.....	2.80	0.80	2.00	0.84	1.64	1.00	0.65
TiO ₂	1.30	1.06	1.20	1.18	1.24	1.30	0.78
P ₂ O ₅	0.15	0.16	0.14	0.16	0.14		
S.....	0.01		0.01				
CuO.....	0.026	0.011	0.003	0.005	0.002		
MnO.....	0.15	0.15	0.15	0.20	0.18	0.13	0.14
BaO.....	trace	trace	trace	trace	trace	trace	trace
	99.51	100.23	99.97	99.72	99.67	99.65	99.51

Cape Spencer			First Watchung Sheet			Connecticut	
	(8) A-587	(9) A-706	(10)	(11)	(12)	(13)	(14)
Distance from base of flow, ft.	179.0	70.0					
SiO ₂	51.76	51.56	51.77	51.09	50.19	52.37	52.40
Al ₂ O ₃	13.49	13.57	14.59	14.23	14.65	15.06	13.55
Fe ₂ O ₃	1.71	2.00	3.62	2.56	3.41	2.34	2.73
FeO.....	7.42	7.80	6.90	7.74	6.96	9.82	9.79
MgO.....	9.05	8.26	7.18	7.56	7.95	5.38	5.53
CaO.....	11.62	10.80	7.79	10.35	9.33	7.33	10.01
Na ₂ O.....	2.38	2.58	3.92	1.92	2.64	4.04	2.32
K ₂ O.....	0.67	0.69	0.64	0.42	0.75	0.92	0.40
H ₂ O.....	0.60	0.90	2.31	2.67	3.04	2.24	1.67
TiO ₂	0.79	1.14	1.13	1.30	1.13	0.21	1.08
P ₂ O ₅			0.18	0.16	0.18		0.12
MnO.....	0.12	0.12	0.05	0.25	0.07	0.32	0.26
Remainder.....							0.13
	99.61	99.42	100.08	100.25	100.30	100.03	99.99

Analyses 1 to 9 by M. F. Connor; 10 to 12 by R. B. Gage (*New Jersey Geological Survey, Annual Report for 1907*, p. 148); 13 by J. H. Pratt (*Bulletin No. 6, Connecticut Geological Survey*, p. 184 (1906)); 14 by W. F. Hillebrand (*Annual Report, U. S. Geological Survey*, vol. xxi; Part 3, p. 77 (1901)).

1 to 4 Fort Grunt flow at Cape d'Or. In core C the thickness of the flow is 132 ft. The top is everywhere eroded away.

5 to 9. Cape Spencer flow at Cape d'Or 5 is 88 ft. below the top of the flow in core C; 6, 167 ft. below the top in core A; 7, 272 ft., in A; 8, 377 ft., in A; 9, 486 ft., in A, and 70 ft. from the base.

10 to 12. First Watchung sheet in Hartshorn's quarry, near Springfield, N. J. 10 represents the "upper gray layer;" 11, the "middle black layer;" 12, the "lower gray layer."

13. Main sheet basalt, Meriden, Conn.

14. Main flow of basalt, Pine Hill, South Britain (Pomperaug), Conn.

In the Watchung analyses (10 to 12), from the top downward there is an increase in magnesia and lime, a decrease in silica, and toward the center an increase in lime and a decrease in potash and ferric iron. An increase with depth in lime and a decrease in soda were found in the Keweenaw rocks.

The combined result shows that there is an increase downward in the percentage of ferrous iron, magnesia, lime, and perhaps potash; a decrease in soda. The change in the percentage of silica is slight. Comparing the chemical composition of the minerals, which have been shown by the microscopic measurements to vary in amounts in different parts of the flow, with the data derived from an examination of the chemical analyses, a close agreement is seen. A separation of feldspar, with a composition of

4.6 per cent. Na_2O , 12.4 per cent. CaO , 20 per cent. Al_2O_3 , 59.2 per cent. SiO_2 , to the top; as against olivine, with 57 per cent. MgO and 43 per cent. SiO_2 , or pyroxene, with 6 per cent. Fe_2O_3 , 18 per cent. FeO , 13 per cent. MgO , 11 per cent. CaO , 3 to 5 $\frac{1}{2}$ per cent. Al_2O_3 , 48 per cent. SiO_2 , toward the bottom, must cause the MgO and Na_2O to vary more than the alumina or silica.

The olivine appears in the thin sections mainly near the top of the flow, although in the Palisade sheet there is, as shown by Lewis, a concentration at the base. Furthermore, the olivine tends to sink in a flow just as in an intrusive body. The discrepancy is explained by the work of Bowen and Andersen⁷ as a result of their recent experiments which show that olivine that has crystallized out above $1,557^\circ$ may be partly or wholly resorbed during the normal course of crystallization simply as the result of cooling, but that this resorption may fail to take place if the cooling is very rapid as it would be near the top of a flow. Also Bowen has shown⁸ that differentiation in silicate liquids may take place by the rising and sinking of crystals. His experiments show that both olivine and pyroxene tend to sink, and that with an excess of silica pyroxene crystallizes first. After the crystallization of the artificially prepared liquid, the upper part consists of pyroxene and free silica, the lower part of pyroxene and olivine. In the basalt flows there is correspondingly more silica at the top of the flow than at the base. It will be remembered that at a range of from $1,890^\circ$ down to $1,387^\circ$ Bowen found forsterite (olivine) to crystallize out, but that at about $1,557^\circ$ it changed to clinostatite (pyroxene).

It may be suggested as a possible reason for the slight accumulation of olivine as compared with the Palisade intrusion, that the latter came to rest when at a higher temperature and lingered at a temperature above $1,557^\circ$ (or the corresponding temperature for an iron-bearing magma like these) much longer than the flow so that olivine could form and could settle out, while in the flow the motion and consequent stirring continued until the whole had reached a lower temperature, but then remained rather long at temperatures at which the particular olivine present inverted to pyroxene and could sink as pyroxene.

This supposition is supported by two additional lines of argument. First, a sinking of pyroxene would make much less difference than a sinking of olivine in a rock which, as the margins show, had originally about 50 per cent. silica. Secondly, the broad zone of relatively uniform

⁷ N. L. Bowen and O. Andersen: The Binary System MgO-SiO_2 , *American Journal of Science*, vol. xxxvii, p. 499 (1914); N. L. Bowen: The Ternary System; Diopside-forsterite-silica, *Idem*, vol. xxxviii, p. 264 (1914); O. Andersen: The System Anorthite-forsterite-silica, *Idem*, vol. xxxix, p. 453 (1915).

⁸ Crystallization-Differentiation in Silica Liquids, *Idem*, vol. xxxix, pp. 175 to 191 (1915).

coarsest grain in the Palisade diabase compared with the narrow zone at Cape d'Or (Fig. 3) indicates clearly that the former was much farther above consolidation temperature than when it came to rest, as has been shown elsewhere by the senior author.

Conclusions

The nature of gravitative differentiation in extrusive rocks has been investigated with the aid of volumetric and chemical analyses, and determinations of specific gravity and grain. The results are in accord and show a concentration of the leucocratic, felsic constituents at the top of the flow; the melanocratic, mafic constituents at the base. The quickly chilled top and bottom of the flow show approximately, when free from alteration, the original composition of the magma.

The grain of the rock in thick flows varies with the depth and is greatest just below the center, where cooling takes place slowest. The grain is difficult to determine in many cases on account of the presence of several generations of crystals, due to convection currents and the subsidence of partially cooled upper portions of the flow; in other cases, as at the top and bottom, on account of the presence of phenocrysts with a rather glassy ground-mass.

The specific gravity determinations have been made in large part by a simple and rapid system of measurements in the case of drill-cores of uniform diameter and a length of over 10 cm. This method has been checked by determinations in the usual manner, and found to be of sufficient accuracy to be of much practical use.

The results of the investigation of differentiation are found to agree with those in experiments recently made by Bowen and Andersen on artificial solutions; that heavier minerals, as olivine and pyroxene, tend to sink while very light minerals may rise.



MARCH, 1916



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MANUSCRIPTS FOR THE ARIZONA MEETING OF THE INSTITUTE

The next meeting of the Institute, the 113th meeting, will be held in Arizona in the latter part of September, 1916. All papers to be presented at this meeting must be published in the September Bulletin or previously. Manuscripts must, therefore, be in the hands of the Secretary of the Institute not later than July 1. In case manuscripts are given by authors to members of technical committees, or others, they must be in the hands of such persons sufficiently in advance so that they may be actually in the hands of the Secretary of the Institute by the date mentioned.

In the past, the Secretary has received urgent requests for special consideration regarding manuscripts that come in after the closing date. In view of the obvious unfairness of this, the Committee on Papers and Publications has instructed the Secretary not to accept for presentation at the Arizona Meeting any papers received later than July 1, 1916.

LOUIS DAVIDSON RICKETTS, PRESIDENT

The following record of the important responsibilities already shouldered by Dr. Ricketts will give some indication of the trust and confidence reposed in him by his colleagues. The Institute is to be congratulated that a man who is already so much engaged should be willing to assume the further responsibility of guiding the Institute's Ship of State during the ensuing year.

Louis Davidson Ricketts, who is a brother of Professor Palmer Chamberlaine Ricketts, President of Rensselaer Polytechnic Institute, was born at Elkton, Md., Dec. 19, 1859, was graduated from Princeton University in 1881 with the degree Bachelor of Science, chosen a Fellow



LOUIS DAVIDSON RICKETTS

in Chemistry and W. S. Ward Fellow in Economic Geology, and after two years' study given the degree Doctor of Science.

Following the completion of his work at Princeton, Dr. Ricketts went to Colorado and started to work as a mine surveyor. For the 15 years following, his time was chiefly occupied in reconnaissance work, geological work and mine examination.

From 1887 to 1890 Dr. Ricketts was Geologist for Wyoming, and at the end of that period transferred his operations to the Southwest, where he has since been steadily engaged in large mining projects. He was identified with the acquisition of the property now owned by the Moctezuma Copper Co., a subsidiary of Phelps, Dodge & Co., at Nacozari, Sonora, Mex. From 1899 to 1901 he was General Manager of the property and during his administration the concentrator and reduction works were completed and the mines put on a dividend-paying basis.

While Dr. Ricketts has had extensive experience in mine examination and management, identified with most of the large and prosperous mines of the Southwest, his most important work has undoubtedly been in the construction of large modern smelting and concentrating plants. All of the plants erected by him have been successful and have brought about great decrease in the cost of handling the ores.

Dr. Ricketts designed his first large concentrators in 1897, when he installed one each for the Detroit Copper Mining Co. at Morenci, Ariz., and the Moctezuma Copper Co. at Nacozari, Mex. These plants had a capacity of 400 tons per day each and were among the first to adopt all-steel construction, Dr. Ricketts being in personal charge of their design and erection.

Upon leaving the Moctezuma Copper Co. in 1901, Dr. Ricketts went to Globe, Ariz., and there undertook the construction of a surface plant and the reopening of the mines of the Old Dominion Copper Mining & Smelting Co. He took this property when it was almost wrecked, and under his administration it was put on a sound, producing basis. For the first time in its history it was made into a property of undoubted value as a dividend payer, this being shown by the rise in its stock value, which advanced without artificial stimulation from \$4.50 to \$65 per share.

In 1903 Dr. Ricketts accepted appointment to the position of Consulting Engineer to the Cananea Consolidated Copper Co. He took absolute charge of the design and construction of the company's new concentrator. Upon the completion of his work he went to Europe and investigated modern engineering practice in the Old World.

Returning to the United States in 1905, Dr. Ricketts, utilizing the knowledge gained in Europe, constructed a large modern coal-washing plant for the Dawson Fuel Co. at Dawson, New Mexico.

In 1907 Dr. Ricketts became identified with the Cananea Consolidated Copper Co. as President and General Manager, and during his administration the works of the company, with the exception of the concentrators, were completely overhauled and rebuilt, and placed upon a profitable basis. He devotes the greater part of his time to the direction of the company's affairs, but in addition to this, he has been in demand by most of the large mining interests of the Southwest in the capacity of Consulting Engineer.

From his first entry into the Southwestern field until 1907, Dr. Ricketts has acted in an advisory capacity to the great Phelps Dodge interests. He was chosen Consulting Engineer for the Calumet & Arizona Copper Co. in 1911, advising it in the design and construction of a great smelting plant at Douglas, Ariz. In 1911 he accepted also the post of Consulting Engineer with the Arizona Copper Co., Ltd., of Clifton, Ariz., and immediately took full charge of the design and construction of a new smelting plant. He also re-designed and enlarged the company's concentrators at Clifton. Dr. Ricketts is also Consulting Engineer for the Anaconda Copper Mining Co., the Inspiration Consolidated Copper Co., the International Smelting and Refining Co., and the New Cornelia Copper Co.

An important part of his work since 1906 has been that done for the Inspiration Consolidated Copper Co., for which organization he has acted as Consulting Engineer in the development of its mines, in the building of its concentrator of 14,000 tons daily capacity, in which flotation of

copper ores was introduced for the first time on a large scale, and in the construction of a smelting plant having a capacity of 16,000,000 lb. of copper a month.

During the past 12 years Dr. Ricketts has, in construction of works alone, directed the expenditure of between \$35,000,000 and \$40,000,000 and has been identified with, and in no small degree responsible for, the great decrease in unit costs and the great increase in recoveries which has so greatly extended the reserves of commercial copper ore in the United States. At the present time he is identified with a process for leaching oxidized copper ores and recovering copper from the resulting solution by electrolytic precipitation, for the New Cornelia Copper Co., Ajo, Ariz. This company is about to erect an initial plant of 4,000 tons daily capacity.

A recent proof of the value of Dr. Ricketts' advice as consulting mining engineer was in connection with the property of the United Verde Extension Mine, at Jerome, Ariz. It was the belief that the only valuable ground in the district was that of the United Verde Copper Co., and that the United Verde Extension property was of no value. After examination of the latter, Dr. Ricketts advised its development. Today it ranks as one of the large mines of the State.

Dr. Ricketts is the author of "The Ores of Leadville and Their Modes of Occurrence," 1883; and "Geological Reports of the Geologist of Wyoming," 1888, 1890, and various papers for technical societies and periodicals. His paper entitled "Experiments in Reverberatory Practice at Cananea, Mexico," secured for him the gold medal of the Institution of Mining and Metallurgy of Great Britain for the year 1910.

Dr. Ricketts is extremely active in the affairs of the Southwest and is interested in various financial and development projects. He is President and Director of the Morenci Water Co., Vice-President and Director of the Gila Valley Bank & Trust Co., Director of the Bank of Bisbee, Bisbee, Ariz., and President of the Tucson, Cornelia & Gila Bend R. R. Co.

He received a medal at the Panama-Pacific Exposition as Arizona's "most distinguished citizen."

From the above account of the activities of Dr. Ricketts it is not hard to believe the statement that he is a stayer for work, and a man of great decision of character. He is a thoroughly democratic citizen, unostentatious, approachable by the most humble miner and always ready to listen to a mining proposition presented by a prospector. The following anecdote illustrating some of these characteristics: Once the Doctor was in the hills examining properties for the Phelps-Dodge interests, and drove by team to Benson, Ariz., wearing his "digging clothes." When he attempted to board Mr. Douglas' private car which had been sent for him, he was met by the porter who refused him admittance, saying the car was a private one and had been sent to bring in Dr. Ricketts and that he had orders to admit no ordinary passengers; the Doctor had to secure someone to identify him before the porter would admit him.

His friends in the Southwest say that the Doctor is the one man in the State of Arizona, connected with and representing large corporations, against whom no unfriendly word has been directed by workingmen or labor agitators. In fact, the workingmen feel that they owe more to Dr. Ricketts than to any other individual, since it is largely through his work that the important mines of the Southwest have been developed, giving employment to thousands of men.

The esteem with which Dr. Ricketts is regarded is shown by the terse

statement of Judge Sutter; "The people of the Southwest believe that no other man has done as much to promote and build up the mining industry as has Dr. Ricketts. Scarcely any mine of importance in Arizona and Northern Mexico has been developed without his assistance."

In a quiet way Dr. Ricketts contributes largely to charity; many old-timers who have met with ill luck have received help from him.

A man of broad vision, a forward-looking man, a doer, such is the new president of the Institute.

We are indebted to the *Press Reference Library*, Judge Fred Sutter, Bisbee, Ariz., and Prof. P. C. Ricketts, Troy, N. Y., for the material used in the preparation of this biography.

ADDRESS OF PRESIDENT, W. L. SAUNDERS, ANNUAL MEETING, NEW YORK, FEB. 15, 1916.

The Institute is at present in sound condition professionally and financially. During the past year 546 new members were elected, the total membership now numbering 5,221. The Treasurer's Report shows that the expenses of the Institute are met by its income. Your Directors have thought it wise to issue a referendum to the members asking for a vote upon the suggestion that the annual dues be raised from \$10 to \$12. This moderate increase is proposed not because the Institute needs it at present but to insure a continuation and increase in its activities and usefulness, and to provide a reserve fund for future emergencies. In well-conducted organizations, such as clubs, it is a wise and safe course to so adjust the dues of members that the total receipts from this source will about equal the annual expenditures, the initiation fees being used for reserve purposes. Our dues alone do not at present meet the expenses. This, I think, is the whole situation, so far as this question is concerned. Expenses might be reduced in some measure, but they cannot be materially reduced without affecting the integrity and usefulness of your Institute.

Nothing of consequence in our internal affairs has occurred during the past year except, perhaps, the establishment of the Arizona Section and the prospect now assured of a Nevada Section.

During recent months there has been a marked change in the relations of the Institute to its fellow societies and to the Government of the United States. That change is one of coöperation. The Secretary of the Navy has appointed two of your members to the Naval Consulting Board to coöperate with members of 10 other scientific organizations. This Board is now organized, and through its work your Institute is brought in close touch with the activities of the other societies and with the United States Navy. The President of the United States has requested us to recommend to the Government one of our representatives in each State in the Union to act in collaboration with a member in each State from the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers and the American Chemical Society, thus organizing a directorate of engineers for each State, that will conduct, through the members of each of the five societies referred to, living in each State, a campaign of industrial preparedness. No more important step than this, it seems to me,

has been made in the history of your Institute. The work which these 48 Boards are expected to do will be directed by a committee of the Naval Consulting Board, and the far-reaching effect and usefulness of this work can hardly be over-estimated. We hear a great deal about preparedness. Many plans have been suggested for increases in the army and navy of the United States and much difference of opinion exists on the subject. As far as I have observed, there is no difference of opinion on the question of industrial preparedness, which means a coördination and coöperation with the Government of our mines, mills, works, and factories, so that in the event of trouble they may be in a position to respond promptly, energetically, and with full force, to the needs of the nation. Even as a peace measure such an organization is desirable. This country is the largest nation in the world industrially, yet our industries are working more or less at cross purposes, are not in touch with each other or, with the Government. Such contact as has existed in the past between business and Government is a special, not a general, contact. Certain industries have had the ear of the Government, while others have not. Nor is the Government at present in a position to feel the pulse of the great industrial strength of the United States in peace or war. In this and in many other ways a general contact will not only be beneficial to the Government but should prove of whole-some advantage to our American industries.

The engineer has been called upon to take this important step. Who is better fitted to do it and do it well? Mind you, it is not the mining engineer alone, but the civil, the mechanical, the electrical and the chemical engineers, coöperating as a unit. We have heard a great deal about the desirability of coöperation among engineers, but so far it has been mainly academic. Here we have a practical fulfillment of our desires and an opportunity the importance of which can scarcely be measured. It is no less important to the engineer and to the whole profession he represents than it is to the industrial strength and prosperity of the nation. The engineer is essentially a man of action, an executive, an administrator, not a mere scientific worker, operating behind closed walls in a dusty laboratory or over a drawing board. His place is out in the middle of the road, with his coat off, leading men, and initiating and directing measures of usefulness to the whole people. We have heard our profession defined as one which uncovers the hidden forces of Nature and puts them at the service of mankind. We have done a good deal in the line of uncovering these hidden forces, but have been very slow in our activities in placing the things revealed in useful operation. Modesty is not only a characteristic but a fault among engineers. Some consider it undignified to get what they call notoriety, but it seems to me that there is a difference between notoriety and reputation; the way to get reputation is to do things, not under a bushel, but in the light, so that the world may know and the full measure of benefit may be derived by inspiring public confidence not alone in what has been achieved, but in the personality of the individual who is responsible for the achievement.

Your Institute, like other long established scientific bodies, has an integrity beyond reproach. It does not work for money but for scientific advancement. Let its usefulness be extended to broader fields and let us resolve to make this organization, which has now been called for by the President of the United States, a useful and permanent force in our whole national life.

NAVAL CONSULTING BOARD

At a meeting of the Board of Directors, held Jan 21, 1916, the following letter from Woodrow Wilson, President of the United States, was read:

MR. W. L. SAUNDERS, *President*,
American Institute of Mining Engineers,
New York City.

January 13, 1916.

My Dear Sir:

"The work which the American Institute of Mining Engineers has done through its members on the Naval Consulting Board is a patriotic service which is deeply appreciated. It has been so valuable that I am tempted to ask that you will request the Institute to enlarge its usefulness to the Government still further by nominating for the approval of the Secretary of the Navy a representative from its membership for each state in the Union to act in conjunction with representatives from the American Society of Mechanical Engineers, the American Society of Civil Engineers, the American Institute of Electrical Engineers, and the American Chemical Society, for the purpose of assisting the Naval Consulting Board in the work of collecting data for use in organizing the manufacturing resources of the country for the public service in case of emergency. I am sure that I may count upon your cordial coöperation.

With sincere regard,

Cordially yours,

(Signed) Woodrow Wilson."

Thereupon, W. L. Saunders, Philip N. Moore, and Bradley Stoughton were appointed a committee, with power to act, in accordance with the request contained therein.

RESOLUTION UPON MEMBERS KILLED IN MEXICO

At a meeting of the Board of Directors, held Jan. 21, 1916, the following resolution was adopted:

Resolved, that this Board has learned with indignation and sorrow of the unprovoked and brutal murder of 18 American citizens on January 10, in the State of Chihuahua, Mexico, and laments especially the death of Messrs. C. R. Watson, C. A. Pringle, H. C. Hase, and W. J. Wallace, who were members of the American Institute of Mining Engineers. As these men and their companies were engaged in the lawful prosecution of their work, we trust that nothing will be allowed to prevent or delay appropriate action by our Government concerning the outrage by which they lost their lives;

Resolved, that the sincere sympathy of this Board and of all the members of the Institute is extended to the families and friends of Messrs. C. R. Watson, C. A. Pringle, H. C. Hase and W. J. Wallace.

And be it further resolved, that a copy of this resolution be sent to the Secretary of State of the United States, be published in the Institute's *Bulletin*, in the press, and be sent to the families of the deceased members.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period Jan. 10, 1916 to Feb. 10, 1916:

William Rettie, Yonkers, N. Y.
R. V. Norris, Wilkesbarre, Pa.
Fred MacCoy, El Oro, Mex.
Reno H. Sales, Butte, Mont.
R. T. White, Batourn, Russia.
David T. Day, Washington, D. C.
H. M. LaFollette, LaFollette, Tenn.
F. N. Speller, Pittsburgh, Pa.
Robert W. Hunt, Chicago, Ill.
B. E. Fernow, Toronto, Ont., Can.
A. F. Bennett, New York City.
J. C. Ralston, Spokane, Wash.
George J. Lyon, Schenectady, N. Y.

Wallace E. Pratt, Manila, P. I.
T. Skewes Saunders, El Oro, Mex.
M. L. Requa, San Francisco, Cal.
R. W. Schultz, Ravenna, O.
H. J. Brown, Chester, Pa.
A. M. Van Rensselaer, New Rochelle, N. Y.
Norman C. Stines, Russia.
Howard R. Hughes, Houston, Tex.
C. F. Schnepf, Ridgefield Park, N. J.
George L. A. Thomson, Elizabeth, N. J.
Jasper T. Robertson, Chicago, Ill.
J. B. Tyrrell, Toronto, Ont., Can.
H. G. Officer, New York City.

Jesse C. Porter is now superintendent of the San Fernando Mine, Hoyo de Manicaragua, Santa Clara Province, Cuba.

Frank W. Tickner has been made general manager of the Valley Mould & Iron Co., Sharpsville, Pa.

A. A. Turner is now manager of the Daggett Reduction Co. and Goldstone Mining Co., Barstow, Cal.

B. G. Slaughter has resigned as vice-president and chief engineer of the Canadian Copper Co. to accept the position of vice-president of the Tennessee Copper Co., Copperhill, Tenn.

Nathan O. Lawton is manager of explorations of about 10,000 acres of land in Hancock County, Tenn., with headquarters at Sneedville, Tenn.

Robert M. Raymond has been appointed Professor of Mining at the Columbia School of Mines, filling the chair made vacant last year by the retirement of **Henry S. Munroe**.

Charles F. Rand was elected president of the United Engineering Society at the Annual Meeting held on Jan. 27, 1916.

Thomas T. Read has joined the engineering staff of the New Jersey Zinc Co., with headquarters at 55 Wall St., New York.

Robert S. Lewis, of the University of Utah, has returned from an extended trip through Arizona and New Mexico, having visited the leading copper mines of the Southwest.

F. G. Clapp, managing geologist of The Associated Geological Engineers, has returned from the Oklahoma fields.

H. A. J. Wilkens and **W. B. Devereux, Jr.**, announce that they have formed a co-partnership for the purpose of carrying on the general business of consulting mining engineers and mine managers, under the firm name of **Wilkens and Devereux**.

Wallace E. Pratt, has resigned his position as Chief of the Division of Mines, Bureau of Science, Government of the Philippine Islands and is engaged in eastern Mexico for private interests.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Mine and mill accountant for Mexico. Knowledge of Spanish desirable. Salary \$150 to \$200 gold per month with living quarters. No. 60.

Master mechanic familiar with mine and cyanide plant machinery for Mexico, \$200 gold per month and living quarters. No. 61.

Electro-metallurgist for production of ferro-alloys, salary \$200 to \$400 depending on experience. No. 62.

Man familiar with ferro-tungsten products of electric furnaces to draw up specifications for machine tools, to watch these tools in service and revise specifications accordingly. Salary \$150 to \$200 per month. No. 63.

Man to take entire charge of the sales of a steel company making electric alloy steels in the shape of ingots, billets and bars. To the right man an opportunity will be offered for taking a leading position in the management of the company and of participating in its results. No. 68.

Chemist wanted by Electric Steel concern in the East. Must be fully acquainted with alloy steel analysis. Send full particulars and references. No. 73.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, graduate engineer, aged 44, with over 22 years' experience in connection with the development of large coal and coke properties in different parts of the country, desires position as general manager or general superintendent of large bituminous coal producer, or will take an interest and the management of a smaller operation. No. 258.

Member, technical graduate in Mining, Chemistry and Metallurgy, Post-graduate, Royal School of Mines, Freiberg, Saxony, Germany and Columbia School of Mines. Age 37, 14 years' experience in teaching mining metallurgy, chemistry and assaying. Available now. Willing to go anywhere. No. 278.

Member, graduate of Columbia School of Mines, with 10 years' experience in the largest metal mines of the West as examining engineer, geologist, and business representative, desires position as geologist, mine superintendent or assistant to consulting engineer. No. 280.

Member, aged 35, married, technical education, 15 years' experience as superintendent, mine surveyor and assayer, mostly of copper, lead, zinc, and silver mines in Scandinavia and Mexico. Desires position with big mining company or as superintendent of small mine. Experienced bookkeeper; can guarantee correct surveying; fluent Spanish; moderate salary; best references from former employers. No. 281.

NEW YORK SECTION

*Executive Committee,*DAVID H. BROWNE, *Chairman,*JOHN H. JANEWAY, *Vice-Chairman,*F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.P. A. MOSMAN, *Treasurer,*

LEWIS W. FRANCIS, BENJAMIN B. LAWRENCE.

At a meeting of the New York Section, held at the Machinery Club on Jan. 19, 1916, the topic for the evening was The Problems of the Naval Consulting Board, on which board are four prominent members of the Institute.

The speakers of the evening were W. L. Saunders, president of the Institute, Dr. L. H. Baekeland, and Commander A. B. Fry, chief of staff of the New York Naval Militia.

After Chairman Browne had laid before the meeting certain business matters, he introduced President Saunders, who presided during the speech making and delivered the first speech. The letter from President Wilson to Mr. Saunders and the presidents of the other national engineering societies, inviting their coöperation in a scheme for mobilizing, as it were, the industrial capacity of this country, had just been made public (see a previous page of this *Bulletin*) and President Saunders described in considerable detail the origin, purposes, and scope, of the work to be undertaken in this direction.

Each of the five national scientific societies is to select a representative from its resident members in every State of the Union, that is, the mining engineers will have 48 representatives, likewise the electrical engineers, the chemists, the civil engineers, and the mechanical engineers. Thus, there will be a representative body or board of five members in every State, one from each of the five societies or institutes. Each committee will investigate the manufacturing and producing possibilities in its own state, and will gather information as to the capacity of every industry which may be capable of supplying things that are needed for the sinews of war.

Industrial preparedness means something more than mere capacity to make shells. The field covers food, clothing, hospital equipment, motors, animals, telephones, telegraph and railway accommodations, and a myriad of small details.

After this information is compiled by these representatives, selected solely for their technical talent, and the data are classified and digested, it is the intention to have the government place small orders for various kinds of war material, from blankets to ammunition, with various plants throughout the country. The contract will not be large, and the profit will not be great, the purpose being to get the manufacturers in touch with the government, to familiarize themselves with the type of material required by the government, to provide themselves with jigs, templates, and other equipment necessary for such work, to get them supplied with blueprints, drawings, etc., and, in a word, to familiarize themselves with the details of such articles as they would be called upon to turn out in large quantities if war should come. All of the work by the committees will be done without remuneration to the various members, and,

furthermore, they will all stand all of their own expenses during the investigation. It is therefore a purely patriotic proposition, free from even the remotest suggestion of politics, and is probably the greatest stride toward preparedness which this country has yet made, because no matter how well drilled or how brave, or how efficient an army may be, it is absolutely dependent upon the supplies and ammunition furnished.

Mr. Saunders then introduced Dr. L. K. Baekeland, who spoke at length on the production of nitric acid, which is a *sine qua non* in the manufacture of war explosives. Nitroglycerin, the basis of dynamite, is made by treating glycerin with nitric and sulphuric acid. Smokeless powder is trinitrated cellulose; *t.n.t.* is trinitrotoluol; and picric acid is trinitrated phenol. Nitric acid is at present manufactured from the nitrates that are procured from Chile, which has the only commercial body of nitrate in the world known at the present time. All of this material is shipped from only one available seaport so that the dependence of the United States upon the maintenance of this open port and clean commercial relations between this port and the United States is evident. Nitrogen fixation, the procuring of nitrogen from the air for nitric acid manufacturing purposes, is a commercial process under some conditions abroad, but so far in this country it has not been a commercial success, largely on account of the cost of power, the consumption of which in the process is very great. At Niagara Falls, nitrogen is recovered by the cyanamid process, by the treatment electrically of calcium carbide. This process, although technically successful, uses power which costs \$18 to \$19 per horsepower-year, while power for similar work in Norway costs only about one-third of this. Dr. Baekeland said that the Germans four or five years ago withdrew all of their heavy capital investment from such Norway plants and concentrated their efforts on local industries, with the result that they have developed nitrogen fixation processes that are supplying them with this necessary requisite for explosives, which, on account of the blockade, they are unable to get from Chile.

Commander A. B. Fry presented valuable information regarding foreign naval reserve systems as compared with the systems and conditions of this country; manifestly Great Britain and Germany are far in advance along these lines. England has, in addition to the Royal Navy, a Royal Fleet Reserve, a Royal Naval Reserve, a Royal Naval Volunteer Reserve, also the British Coast Guard Service. The Royal Fleet Reserve consists of enlisted men, there being no officers in this body; the Royal Naval Reserve is not an organized body, officers and men being called on individually for training or service; the Royal Naval Volunteer Reserve comprises five companies of 100 men each. Each Division is commanded by a Lieutenant, with two sub-Lieutenants, a detail of Junior Officers and Petty Officers from the Royal Navy. In addition to these reserves, there has been organized, to meet the exigencies of the present war, an Auxiliary Force comprising trawlers, fishermen, other longshoremen not serving in the Royal Naval Reserve, and yachtsmen or amateur boatmen not serving in the Royal Naval Volunteers.

The last available figures before the war showed that the British Navy had about 8,500 officers and 150,000 Bluejackets and Marines; the Royal Fleet Reserve 20,000 men (no officers); the Royal Naval Reserve 2,000 officers and 25,000 men; the Royal Naval Volunteers 200 officers

and 5,000 men. Normally the Coast Guard had 350 officers and 4,000 men. These numbers doubtless have been greatly increased for war service, and it was recently estimated by an officer in Washington who had some facilities and opportunities for observation in England that there were 60,000 men serving in various branches of the British Auxiliary Fleet and Naval Reserve. The British Naval Reserve System is a volunteer service. To some of the officers and men patriotism is the impetus; others are attracted by the retaining fees and pensions after the period of service. Many of the Merchant Officers are held by the honor of serving in the Royal Naval Reserve and the practical advantages attendant, such as official precedence and the right to fly the blue ensign.

Anent the German system of organizing and maintaining forces; strictly speaking, there is no volunteering under the German method. Every man, on reaching the prescribed military age, appears before the authorities for examination. The Navy Department annually makes requisition on the War Department for recruits who are drawn from all districts of the Empire. Service is obligatory, without the payment of annual retainer to the men of the Reserve, though if called to colors in peace or war, they receive the pay of their grades. Prior to the war it is believed that there were about 6,000 officers and 75,000 men in the active Navy, and about 2,000 Reserve Officers and 150,000 reserve men available. The proportion of officers to men was considerably higher in Germany than in Great Britain.

The United States boasts a Volunteer Naval Militia which, with the exception of the Russian Volunteer Fleet, stands alone.

The United States Naval Militia at present is organized in 22 States, the number in each varying; the following figures illustrate the proportion: New York, 1,500, California 850, Massachusetts 700, Illinois 650, Florida 100. The total number of commissioned officers is 600, total number of enlisted men about 8,000.

CALIFORNIA SECTION

Executive Committee

T. A. RICKARD, *Chairman*,

W. H. SHOCKLEY, *Vice-Chairman*,

C. E. GRUNSKY, *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.

E. A. HERSAM,

H. W. YOUNG.

The Annual Meeting of the San Francisco Section was held on Jan. 11, 1916. After the dinner at the Hotel Oakland, 50 members and guests availed themselves of an opportunity to see several types of flotation apparatus in operation at the laboratory of Charles Butters. Mr. Butters demonstrated the methods of producing different kinds of flotation froth and discussed informally the flotation methods now in use in his laboratory, emphasizing the fact that present knowledge of the underlying principles of the process is very limited. The visit to the laboratory was of much interest, and especially to the men who have not previously witnessed the operation of flotation processes.

Following an informal discussion and refreshments, the regular meeting was held. T. A. Rickard, Chairman of the Arrangements

Committee, presided at the meeting, which resulted in the election of the Executive Committee noted above. The present membership of this Section is 232. C. E. GRUNSKY, JR., *Secretary*.

CHICAGO SECTION

Executive Committee,

R. W. HUNT, *Chairman*,
J. A. EDE, *Vice-Chairman*,
H. W. NICHOLS, *Secretary*, 5213 Blackstone Ave., Chicago, Ill.
F. K. COPELAND, G. M. DAVIDSON.

The Chicago Section met on Friday, Jan. 14, 1916, at the Chicago Engineers' Club. Robert W. Hunt, Chairman, presided at the meeting, at which 46 members and guests were present. Dr. John A. Mathews, president of the Halcomb Steel Co., Syracuse, N. Y., presented a paper entitled Iron in Antiquity and Today.

HENRY W. NICHOLS, *Secretary*.

MONTANA SECTION

Executive Committee

F. M. SMITH, *Chairman*,
J. L. BRUCE, *Vice-Chairman*,
D. C. BARD, *Secretary*, Box 267, Butte, Mont.
FREDERICK LAIST, W. C. SIDERFIN.

The Montana Section held its Annual Meeting on Feb. 4, 1916, at the Silver Bow Club. An informal dinner preceded the meeting after which the election of officers and other business of the Section followed. D. C. Bard and Chester Steele presented a paper entitled Present Status of Oil and Gas Prospecting in Montana. D. C. BARD, *Secretary*.

NEVADA SECTION

At a meeting of the Board of Directors, held Jan. 21, 1916, upon the written request of 27 members of the Institute resident in Nevada, it was resolved to establish the Nevada Section of the Institute, its territory to include the whole State of Nevada.

PUGET SOUND SECTION

Executive Committee

GLENVILLE A. COLLINS, *Chairman*,
H. L. MANLEY, *Vice-Chairman*,
AMOS SLATER, *Secretary-Treasurer*, Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

The Puget Sound Section met on Jan. 29, 1916, at the Arctic Club. The Chairman read a letter from Secretary Stoughton, relative to the action of the Institute on National Preparedness in connection with a

letter from President Wilson. The following motion, made by I. F. Laucks and seconded by C. F. Lee, was carried: That the Puget Sound Section coöperate with the Institute in all possible ways within its power in giving assistance to the Naval Consulting Board, as suggested in the letter from Secretary Stoughton.

There being no further business to transact, the Chairman presented the speaker, Marcel Daly, a French engineer, who delivered an admirable paper, The Diastropic Theory of Oil and Gas Accumulation.

AMOS SLATER, *Secretary*.

THE MEXICO SECTION

At the meeting of the Board of Directors, held Feb. 15, 1916, upon the request of 40 members of the Institute, residing in Mexico, the Mexico Section was established by consolidating the Mexican Institute of Mining and Metallurgy with this Institute.

THE BOLIVIA SECTION

At the annual dinner of the Institute, held Feb. 16, 1916, a cablegram from President-Elect, L. D. Ricketts, and Past-President, B. B. Thayer, was read, announcing that the Bolivian Local Section had been formed and that dinner was being held at Corocoro. This information was received with enthusiasm by those in attendance at the banquet in New York.

AFFILIATED STUDENT SOCIETY NOTES

Colorado School of Mines Scientific Society held its first meeting on Friday evening, Jan. 14, 1916. H. C. Parmalee gave an interesting talk on the Loss of Cyanide by Hydrolysis.

F. E. BRIBER, *Secretary*.

The School of Mines Society of the University of Minnesota held a meeting on Thursday evening, Jan. 13, 1916, at which Fred A. Jordan gave a talk on Magnetic Concentration of Iron Ores. Mr. Jordan is at present testing the new magnetic concentrator designed at the School of Mines Experiment Station. This apparatus, if successful, will open up an enormous tonnage of ore on the Cuyuna and Mesabi ranges.

ADOLPH DOVRE, *President*.

At the College of Mines, University of Washington, Seattle, 33 mining men, with experience varying from one to 25 years, have enrolled in the 19th Annual Three Months' Short Session. At the annual smoker tendered the Short Session miners by the student branch of the American Institute of Mining Engineers, Capt. R. H. Stretch of the Alaska Bureau, Seattle Chamber of Commerce, delivered an address on The Mineral Belts of Alaska.

COMMITTEE ADVISORY TO THE U. S. BUREAU OF STANDARDS

Copper: F. L. ANTISELL, *Aluminum:* JOSEPH W. RICHARDS,
Lead: HEINRICH O. HOFMAN, *Zinc:* GEORGE C. STONE.
Nickel: DAVID H. BROWNE,

At the semi-annual meeting of the Committee Advisory to the United States Bureau of Standards, Washington, D. C., held Nov. 19, 1915, motion was made and seconded that the Bureau proceed with the work of experimenting to determine the effect of various deoxidizing agents on the copper-tin-zinc (88-10-2) alloy, starting with phosphor copper, which is the one most available and most common in use.

In connection with the matter of practical commercial tests for the *elastic limit*, it was suggested that inasmuch as foreign governments are buying materials on the certifications of the Bureau of Standards, the Bureau should be able to formulate an arbitrary definition of *elastic limit*, thus precluding the possibility of ambiguity of expression and clearly defining the term, so that it would not be possible for one inspector of a manufacturing firm to construe *elastic limit* in one sense, and another inspector of the same firm to accord the term a different construction.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Gift of Aeronautical Society

[The Aeronautical Society has turned over to the Library all of its Library; as soon as this collection can be listed it will be printed in detail in a future issue of the Bulletin. W. P. Cutter, Librarian.]

Mining, Metallurgy and Chemistry

HANDBUCH DER MINERALCHEMIE. Bd. II, No. 8. By C. DOELTER. Dresden, 1915.

LIMITS OF INFLAMMABILITY OF MIXTURES OF METHANE AND AIR. Technical Paper 119, U. S. Bureau of Mines. Washington, 1915.

MANUFACTURE AND USES OF ALLOY STEELS. Bull. No. 100, U. S. Bureau of Mines. Washington, 1915.

MINE-VENTILATION STOPPINGS, WITH SPECIAL REFERENCE TO COAL MINES IN ILLINOIS. Bull. No. 99, U. S. Bureau of Mines. Washington, 1915.

SAMPLING AND ANALYSING FLUE GASES. Bull. No. 97, U. S. Bureau of Mines. Washington, 1915.

LEHRBUCH DER EISENHÜTTENKUNDE. Band I. (Text-book of Iron Metallurgy, for Instruction, Practice, and the Design of Ironworks. Vol. I. The Production of Pig Iron.) By BERNHARD OSANN, Professor at the Royal Mining Academy of Clausthal. Leipzig, Engelmann, 1915. (Gift of Publisher.)

DETAILS OF PRACTICAL MINING. Compiled from the *Engineering and Mining Journal* by The Editorial Staff. First Edition. McGraw-Hill Book Co., Price \$5. (Gift of Publisher.)

[NOTE.—This is the third of a series of books, the first of which dealt with the practical details of mining, and the second with those of milling. The present volume is a continuation of the first. Each volume is made up from the department of the *Engineering and Mining Journal* devoted to its subject, and consists of descriptions by contributors or members of the editorial staff, of notable new methods and devices encountered in the field of practice. The present, like its predecessor, covers three years of the *Journal*, and is, of course, only a selection of the most interesting and valuable items out of a much larger number. Such a collection of professional "news," chosen with discrimination, carefully edited, and arranged and indexed for convenient reference, is a welcome benefit to both student and practitioner. Under the headings, Surface Plant and Operations, Explosives, Rock Drills, Shafts and Raises, Drifting, Stopping, Timber Structures, Hoisting, Lowering, Transporting, Shaft Conveyances, Cars, Track, Safety and Sanitation, Drainage and Ventilation, the reader will find here a host of up-to-date records and suggestions. The work reflects much credit on the editorial staff which intelligently observed, clearly described, discussed, and finally put into this timely and helpful form, facts which progressive mining engineers both need and desire.

Mr. Ingalls, in his preface, ascribes the chief credit for the selection, arrangement, and careful revision of material to Lee O. Kellogg, who is also the author of many articles in the book. It is a pleasure to repeat this praise and recognition. R. W. R.]

METALLOGRAPHY AND HEAT TREATMENT OF IRON AND STEEL. By ALBERT SAUVEUR, Professor of Metallurgy and Metallography in Harvard University and the Massachusetts Institute of Technology. Second Edition (Third Thousand). Sauveur & Boylston, Price \$6. (Gift of Author.)

[NOTE.—The appearance of a second edition of this comprehensive and valuable book will be welcomed by all those interested in metallography in general, as well as those interested only in the heat-treatment of steel and the metallography of iron and steel. Although it is less than three years since this book first appeared, two impressions of one thousand each have already been exhausted. Dr. Henry M. Howe has said that to keep this subject up to date would mean publishing it on Greek kalends. This circumstance alone would justify the publication of a second edition even if the rapid exhaustion of the first edition had not indicated the need. The author has taken the opportunity to bring the whole subject up to date, enlarging especially the sections devoted to apparatus and manipulation, the thermal curves of iron and steel, and heat treatment. All sections of the work have been revised in accordance with the advance in our knowledge and need shown by the use of the first edition, and additions have been made especially under the head of "Heat Treatment and Special Steels." This book has won its place not only as one of the best books ever published on the subject of metallography, but as the most comprehensive and instructive treatment of the special department of iron and steel metallography. B. S.]

LEHRBUCH DER EISENHÜTTENKUNDE. Band I. (Text-book of Iron Metallurgy, for Instruction, Practice, and the Design of Ironworks. Vol. I. The Production of Pig Iron.) By BERNHARD OSANN, Professor at the Royal Mining Academy of Clausthal. Leipzig, Engelmann, 1915. (Gift of Publisher.)

[NOTE.—In this octavo of 668 pages, with 17 plates and 407 illustrations in the text, Professor Osann gives an up-to-date picture of modern blast-furnace practice, especially as it exists in Germany, which will be highly valuable to American metallurgists. In his Introduction, which contains a very interesting historical survey of the subject, after acknowledging the leadership of America in the modern development of the blast furnace, he says, "The case was not altered until, at the beginning of this century, a number of German machine works went forward on their own account, allowing to America only the authority of the initiative. Thanks to their achievements, our blast-furnace chargers, arrangements for unloading vessels, machines for handling the rails in rolling, and rapid traveling cranes, have come to pass. The case has been reversed. Today the Americans are learning from us."

There is doubtless truth in this boast, though it is rather in the use of blast-furnace gases than in the particulars enumerated by Professor Osann, that the Americans have

most to learn. It must be confessed that he does less than justice to our practice and its results; but that is a trifling matter. Nobody in this country needs to buy this book in order to find out what we are doing; what we want is to get the latest reports of what is done in Germany, and there is no doubt that we can study those reports with advantage.

A second volume, treating of the manufacture of wrought-iron and steel, is promised.
R. W. R.]

Geology and Mineral Production

CALIFORNIA. MINES AND MINERAL RESOURCES OF AMADOR, CALAVERAS, TUOLUMNE, COLUSA, GLENN, LAKE, MARIN, NAPA, SOLANO, SONOMA, YOLO, SHASTA, SISKIYOU, TRINITY, DEL NORTE, HUMBOLDT, MENDOCINO, FRESNO, KERN, KINGS, MADERA, MARIPOSA, MERCED, SAN JOAQUIN, STANISLAUS COUNTIES. 5 Vol., San Francisco, 1915.

CANADA. COPPER DEPOSITS OF THE EASTERN TOWNSHIPS OF THE PROVINCE OF QUEBEC, REPORT. Quebec, 1915.

CANADA. PETROLEUM AND NATURAL GAS RESOURCES OF CANADA. Vol. II, Ottawa, 1915.

NEW MEXICO. STATE MINE INSPECTOR. Report 1912, 1915, v, p. 1912. (Gift of State Mine Inspector.)

WEST VIRGINIA GEOLOGICAL SURVEY. COUNTY REPORTS. WYOMING AND McDOWELL COUNTIES, WITH MAP. Wheeling, 1915. By Ray V. Hennen.

[NOTE.—This detailed county report covers one of the principal areas of the great Pocahontas or "smokeless" coal fields of West Virginia, giving a complete account of each coal bed, with analyses, estimates or unmined tonnage, and topographic and structural maps showing the elevation, dip, and strike of the principal coal beds, including the famous No. 3 Pocahontas, covering the Counties of Wyoming and McDowell, the latter leading every other county of West Virginia in the production of coal of the highest grade by several million tons annually. Price, with case of maps, delivery charges paid by the Survey, \$2.50; special combination price with other publications. Extra copies of geologic map, \$1 each, topographic map, 50 c. each.]

Military Engineering

Technique of Modern Tactics. Ed. 2. By P. S. BOND and M. J. McDONOUGH. Menasha, 1915.

General

ENGINEERING INDEX. 1915. New York, 1915.

HAZARDS IN HANDLING GASOLINE. *Technical Paper* 127, U. S. Bureau of Mines. Washington, 1915.

PANAMA CANAL. ANNUAL REPORT OF THE GOVERNOR, WITH MAPS. 1915. Washington, 1915.

SCIENCE OF WORKS MANAGEMENT. By JOHN BATEY. London, 1914.

Trade Catalogues

ALLIS-CHALMERS MANUFACTURING Co., Milwaukee, Wis. Bull. 1636. Hydraulic Machinery, Oct., 1915.

DENVER FIRE CLAY Co., Denver, Colo. Oil flotation process. 8 pp.

GENERAL ELECTRIC Co., Schenectady, N. Y. Bull. No. 47750. Switchboard structural devices and accessories. Dec., 1915.

INTERNATIONAL NICKEL Co., New York, N. Y., Monel Metal, descriptive booklet.

LESCHEN & SONS ROPE Co., St. Louis, Mo. Leschen's Hercules. Jan., 1916.

SCIENTIFIC MATERIALS Co., Pittsburgh, Pa. Apparatus and supplies for the preparation of metal specimens for microscopic investigation. (Booklet.) 16 pp.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Jan. 10, 1916 to Feb. 10, 1916.

- ATWATER, MAXWELL WANTON, Min. Engr. Box 156, Basin, Mont.
BARKDOLL, IVAN HARRY, Supt. of Mines, Old Dominion Copper Min. & Smelt. Co.,
Globe, Ariz.
BEAUDRY, WILLIAM A., Managing Director ... Stewart Mining Co., Kellogg, Ida.
BRIGGS, CHARLES H. Mill Supt., Admiralty Zinc Co., Miami, Ariz.
CHURCH, L. C., Min. Engr. Joplin, Mo.
DAVIS, LLOYD J., Sales Engr. Ingersoll-Rand Drill Co., Joplin, Mo.
DAVIS, SIDNEY H., Mgr. Century Zinc Co., Joplin, Mo.
DUGAN, J. H., Engr. and Contractor. 65 N. Vine St., Hazleton, Pa.
ELLIS, HUBERT INGERSOLL, Min. Engr., Bunker Hill & Sullivan Min. & Cons. Co.,
Kellogg, Ida.
FISHER, THOMAS E., Min. Engr., Care of C. Santos, Ltd.,
42 Rue Commercial, Lisbon, Portugal.
FLEMING, JAMES RUSSELL, Asst. Min. Engr. U. S. Bureau of Mines, Urbana, Ill.
GROSS, LEROY MAURICE, Mining. 431 Riverside Drive, New York, N. Y.
HAINES, P. A. Div. Supt., Doe Run Lead Co., Rivermines, Mo.
HANSON, HENRY, Met. Engr. 852 Clayton St., San Francisco, Cal.
HARDY, THOMAS W., JR., Met., New England Westinghouse Co., Springfield, Mass.
HARRIS, WALTER, Chemist. Old Dominion Min. & Smelt. Co., Globe, Ariz.
HOFFMAN, RAY E. 419 N. 5th St., Hannibal, Mo.
ISHIKAWA, ICHIRO. Asst. Prof., Tokyo Imperial Univ., Tokyo, Japan.
IZOD, EDWIN GILBERT, Cons., Elec., and Mech. Engr., Central Min. & Investment
Corpn., Ltd., & Rand Mines, Johannesburg, Transvaal, So. Africa.
JENKINS, LEMUEL RAY, Asst. Supt., Mammoth Copper Mining Co.,
Mammoth, Shasta Co., Cal.
JENNINGS, ROBERT E., 2d. Genl. Mgr., Hewitt Steel Corpn., Newark, N. J.
KENRICK, JOHN P., Genl. Agent and Chief Engr., Pekin Syndicate, Ltd.,
Peking, China.
LI, KUO CHING, Min. Engr., Wah Chang Min. & Smelt. Co., Ltd.,
Woolworth Bldg., New York, N. Y.
LIANG, HUAN YI, Chief Engr., Wah Chang Min. & Smelt Co., Changsha, China.
LIDBURY, F. AUSTIN, Wks. Mgr., Oldbury Electro-Chemical Co., Niagara Falls, N. Y.
LOW, JOHN CHAMBERS, Min. Engr. Box 2343, Globe, Ariz.
MCMILLAN, W. D., Geol., McIntyre-Porcupine Mines, Ltd., Schumacher, Ont., Can.
MACKENZIE, J. D., Geol., Instructor in Geol.,
Massachusetts Institute of Technology, Boston, Mass.
NISSEN, ARVID E. Foreman, Illinois Steel Co., Gary, Ind.
OMORI, ICHIRO, Prof. of Met. and Assaying,
Kumamoto High Technical School, Kumamoto, Higo, Japan.
RICHARDS, ADELBERT H. Asst. Supt., Amer. Smelt. & Ref. Co., Murray, Utah.
SALADIN, EDOUARD E., Min. Engr. 640 Riverside Drive, New York, N. Y.
SCHULTZ, CASPER I., Mining; Genl. Supt., Moctezuma Copper Co., Nacozari,
Son., Mex.
SMITH, WILLIAM N., Genl. Mgr. Vinegar Hill Zinc Co., Platteville, Wis.
UTTER, GEORGE H., Min. Engr. 455 9th St., Silver City, N. Mex.
VOORHEES, FREDERICK AUSTIN, Met., Mill Supt., Commonwealth Min. & Mill. Co.,
Pearce, Ariz.
WASLEY, WILLIAM A., Cyanide Foreman, The Tigre Min. Co., Esqueda, Son., Mex.
via Douglas, Ariz.
WELLS, ARTHUR E., Met. U. S. Bureau of Mines, Berkeley, Cal.
WILSON, PHILIP D., Min. Engr., Geol., Calumet & Arizona Min. Co., Warren, Ariz.

Junior Members

- BOYD, DANIEL, Student. University of Toronto, Toronto, Ont., Canada.
BRENNEMAN, FREDERICK G. Box 632, Golden, Colo.
BURRAGE, ROBERT H. 38 Conant Hall, Cambridge, Mass.
BYERS, WHEATON BRADISH. 50 Hampden Hall, Cambridge, Mass.
CURRAN, FRANCIS R. 322½ N. Main St., Butte, Mont.

DAVIES, FRED A.	2529 Pleasant Ave., Minneapolis, Minn.
DRAKE, MORRIS CLAIRE, Min. Engr.	North East, Md.
FAUST, GUY C.	617 West Norwegian St., Pottsville, Pa.
FULLAWAY, RICHARD MERLE	Box 518, Golden, Colo.
GARRISON, MURRAY	Box 593, Golden, Colo.
GAUTHIER, CHARLES B.	Box 398, Golden, Colo.
HERR, JOHN CAMPBELL	Box 556, State College, Pa.
HILL, JOSEPH H.	Box 625, Rapid City, So. Dak.
HLEBNIKOFF, KENNETH IVANOVICH	Blagowetchensk on Amur, Russia in Asia.
KAMP, WILLIAM HENRY	Missouri School of Mines, Rolla, Mo.
KROGH, ALVIN THOMAS	2448 23 Ave. So., Minneapolis, Minn.
LAVENDER, HARRISON M.	Colorado School of Mines, Golden, Colo.
LYON, THOMAS T., JR.	834 West Granite St., Butte, Mont.
MCLELLAND, H. L.	Cooksville, Ont., Canada.
MCERODAN, BYRON ALFRED	50 Admiral Road, Toronto, Ont., Canada.
MCHARDY, ROY H., Student	Minnesota School of Mines, Minneapolis, Minn.
NEUSTAEDTER, HAROLD A.,	Missouri School of Mines and Metallurgy, Rolla, Mo.
PLACE, RICHARD G.	661 Lake St., Reno, Nev.
RALPH, WALTER HERBERT	Box 488, Golden, Colo.
REILLY, JOHN H. GAY	Missouri School of Mines, Rolla, Mo.
SANFORD, HAROLD EDMUNDS	Chi Psi Lodge, So. Bethlehem, Pa.
SHERMAN, FRED W., Assayer and Chemist,	Wasp No. 2 Mining Co., Flatiron, So. Dak.
SMITH, FRANK AUGUSTUS	Box 668, Golden, Colo.
SMITH, ELWYN LEANDER	503 Columbus St., Rapid City, So. Dak.
STEELE, CHESTER H.	220 N. Columbia St., Butte, Mont.
WONG, SAMUEL CHUCK	120 N. Montana St., Butte, Mont.
Total Membership, Feb. 10, 1916.	5301

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Jan. 10, 1916 to Feb. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Karl Fredrik Altio, Polefakoy Zavod, Perm Govt., Russia.

Proposed by Edward L. Stenger, E. J. Carlyle, Norman C. Stines.

Born 1877, Ulcaborg, Finland. 1901, Grad., Chem. Engr. 1902, Surveyor, Polytechnical Institute of Finland. 1908, Grad., Univ. of Finland, M. Sc. 1911, Grad., Michigan College of Mines, E. M. 1898, Pitkaranta Copper and Tin Wks., Finland. 1899 and 1900, summer, Asst. Geol., Finland Geol. Survey. 1902-05, Asst. Engr. 1905-08, Engr., Finland Land Survey and Valuation. 1910, Trammner, Champion Mine, Champion Copper Co., Painesdale, Mich. 1910, Helper, Ahmeek Concentration Mill, Ahmeek Min. Co., Hubbell, Mich. 1912-13, Met. and Metallographist, Illinois Steel Co., Chicago, Ill. 1913, Helper, converters, Kennett Plant, Mammoth Copper Min. Co., Kennett, Cal. 1913, Helper, blast furnaces, reverberatories and McDougalls, Garfield, Utah. 1913, Helper, Leonard Mine, Anaconda Copper Co., Butte, Mont.

Present position: 1914 to date; Min. Engr., Sissert Mining District Co., Ltd.

Arthur Ernest Blackwood, New York, N. Y.

Proposed by W. L. Saunders, George A. Howells, W. R. Grace.

Born 1868, Toronto, Canada. 1895, Grad., Toronto Univ., Engrg. Diploma, School of Practical Science. 1896, Appointed Fellow, Elec. Engrg., Toronto Univ.; position of Instructor. 1897, Appointment renewed for second year. During these two years did research work. 1898, Draughtsman, Goldie & McCulloch, Galt, Ont., Canada. 1899, Mining Machinery Salesman, Sullivan Mach. Co., Claremont, N. H.

Present position: 1900 to date; Sullivan Machinery Co.; consulted by mining companies respecting mining methods and equipment.

Frank C. Breyer, Palmerton, Pa.

Proposed by Francis P. Sinn, J. A. Singmaster, George C. Stone.

Born 1887, Baltimore, Md. 1908, Johns Hopkins Univ., A. B. 1910, M. A. 1910-11, Chemical Investigator. 1912, Asst. Chief, Testing Dept., N. J. Zinc Co. (of Pa.).

Present position: 1912 to date; Chief, Testing Dept., N. J. Zinc Co.

Frank P. Brunel, Ajo, Arizona.

Proposed by Charles A. Mitke, L. B. Mitchell, William R. Grunow.

Born 1888, Lens, France. 1902-06, Golden High School, Colo. 1912, Colorado School of Mines, M. E. 1906, Red Metal Min. Co., Butte, Mont. 1906, Survey of Chicago, Milwaukee and Puget Sound R. R. 1909-10, Mining, Adamant Brick Co., Boulder, Colo. 1912, Engrg. Dept., Ray Cons. Min. Co. 1912, Millman, Engrg. Dept., Tonopah Belmont Min. Co. 1913, Calumet & Arizona Min. Co. 1913-14, Engrg. Dept., Shattuck Arizona Copper Co.

Present position: Chem., Ajo Cons. Min. Co.

George Bradford Corless, Joplin, Mo.

Proposed by H. A. Buehler, C. A. Wright, Edward M. Shepard.

Born 1890, Silverwood, Mich. 1903-06, High School, Mayville, Mich. 1906-08, Normal School, Mt. Pleasant, Mich. 1909-13, Univ. of Michigan, A. B., A. M. 1912, Field Asst. Michigan Geological Survey. 1913-14, Field Geol., F. I. Carpenter Syndicate.

Present position: 1914 to date; Geol., Missouri Bureau of Geology and Mines.

José Isaac Corral y Alemán, Havana, Cuba.

Proposed by E. Aguilera, Pedro Aguilera, Eduardo J. Chibas.

Born 1882, Cardenas, Cuba. 1905, Grad., Escuela Especial de Ingenieros de Minas de Madrid, Spain, M. E. 1906, Cuba Public Works Dept., Engineer of inspection of road, Guaos to Cumanayagua, Cuba. 1907, Engineer of survey of road, Cumanayagua to Manicaragua. 1908-09, 1st Asst. Engr., Public Works, Sta. Clara Province.

Present position: 1910 to date; Min. Engr., Office of the Director of Mines & Forestry.

William Elmer Crawford, Pachuca, Hgo., Mex.

Proposed by Hugh Rose, William J. Cox, S. D. Sproat.

Born 1881, Litchfield, Ill. 1898, Grad., High School, Santa Clara, Cal. 1899, Grad., Washburn Preparatory School, San Jose, Cal. 1906, Grad., Stanford Univ. A. B. 1903-06, Student Asst., Chemistry Dept., Stanford Univ. 1906, Asst. Analyst, Reclamation Service, U. S. G. S., Berkeley, Cal. 1906, Teacher of Science, Auburn High School, Auburn, Cal. 1907, Chem. and Assayer. 1908, Night Foreman, mill and cyanide plant, Palmarejo & Mexican Gold Fields, Ltd., Chinipas, Chih., Mex. 1908, Head sampler, mine and smelter; 1909, Smelter Chem., Lustre Min. Co., Magistral, Dur., Mex. 1910, Asst. Mill Supt., Virginia & Mexico Min. Co., Hostotipaquillo, Jal., Mex. 1910-11, Experimental work, Cia. Beneficiadora de Pachuca, Pachuca, Hgo., Mex. 1911-12, Foreman, treatment of sand tailings, Hacienda de Guadalupe, Cia. Beneficiadora de Pachuca, S. A., Pachuca, Hgo., Mex. 1912-13, General experimental and other work, Cia. Beneficiadora de Pachuca, S. A., Pachuca, Hgo., Mex.

Present position: 1914 to date; Mill Supt., Cia. Beneficiadora de Pachuca, S. A.

Theodore M. Daulton, Seattle, Wash.

Proposed by Chester F. Lee, F. A. Hill, I. F. Laucks.

Born 1860, Iowa. 1876-80, Monona, Ia., High School. 1881-87, Quartz Mining, roustabout to chief engr., Silver King Mine, Ariz. 1905-16, Placer Mining in Alaska and British Columbia.

Present position: 1908-16, Mgr., Placer Gold Mines Co., Atlin, B. C.

Elbridge Gerry Deane, Miami, Ariz.

Proposed by B. Britton Gottsberger, F. W. MacLennan, H. P. Bowen.

Born 1879, Buffalo, N. Y. 1898-1901, Michigan College of Mines, B. S., E. M. 1901, Transitman for George R. Sikes, Buffalo, N. Y. 1901-02, Asst. Engr., Oliver Iron Min. Co., Ely, Minn. 1902-05, Engr., Chandler Iron Co., Ely, Minn. 1905-06, Chief Engr., Newport Min. Co., Ironwood, Mich. 1906-07, Asst. Mine Supt., Nevada Cons. Copper Co., Ely, Nev. 1908-10, Chief Min. Engr., Cerro de Pasco Min. Co., Cerro de Pasco, Peru. 1911-13, Chief Min. Engr., Inspiration Cons. Copper Co., Miami, Ariz.

Present position: Slicing Efficiency Engr., Miami Copper Co.

Frank Austin Downes, Aurora, Nev.

Proposed by Arthur Chester Dammann, W. M. Dake, Jr., E. A. Julian.

Born 1888, Trinidad, Colo. 1905-09, High School, Idaho Springs, Colo. 1909-13, Colorado School of Mines, Golden, Colo., E. M. 1913-14, Experimental Dept., Ohio Copper Min. Co., Lark, Utah. 1914, Solution man, Deadwood Mill, Mogollon, N. M. 1914-15, Shift Boss and Refiner, Ernestine Mill, Mogollon, N. M.

Present position: 1915 to date; Solution man, Aurora Cons. Mill.

J. G. Flynn, Miami, Ariz.

Proposed by B. Britton Gottsberger, F. W. MacLennan, H. P. Bowen.

Born 1876, Minn. 1898-1902, Grad., Univ. of Minn., E. M. 1902-04, Engr., Anaconda Copper Co., Butte, Mont. 1904-06, Engr., Wingfield & Nixon, Goldfield, Nev. 1906-08, Supt., Goldfield Cons. Min. Co. 1908-10, Supt., Santa Cruz Min. Co., Sinoloa, Mex. 1910-12, Prospecting.

Present position: Engr., Miami Copper Co.

L. Julio Foster, San Bernardo, Chile, So. Amer.

Proposed by Mark R. Lamb, Berth. Koerting, T. H. Barclay.

Born 1853, Falca, Chile, So. Amer. 1866-67, Lyceum of Curico. 1867-70, National Institute, Santiago. 1870-71, English College, Santiago, Chile. 1871-77, Employed in the Commerce. 1877-84, Genl. Mgr., copper mines, Coquimbo, Chile. 1884-91, Public employee, Gov. of Chile. 1891-1909, Genl. Mgr. (and associate) copper mines, Santiago and Curico. 1909-13, Genl. Mgr., Cia. Estanifera de Llalagua, Bolivia.

Present position: Promotor of mining business.

R. W. French, Goldfield, Nev.

Proposed by K. M. Simpson, George H. Garrey, Willis Lawrence.

Born 1880, Denver, Colo. 1892, Common School. 1905 and 1907, Special course, Univ. of Colo. 1899-1913, Assayer, Chem., Supt., U. S. Red. & Ref. Co., Colorado Springs and Florence, Colo. 1913, Kingman Copper Co., Mineral Park, Ariz.; Chem., Good Luck Mill, Boulder, Colo.; Supt., Dominion Red. Co., Cobalt, Ont. 1914, Met. and Mill Supt., Aurora Cons. Mines Co., Aurora, Nev.

Present position: Met., Goldfield Cons. Mines Co.

John R. Griffith, Niagara Falls, N. Y.

Proposed by C. L. Colburn, Fred H. Bostwick, J. M. McClave.

Born 1882, Pontardawe, Wales. Completed grade school work in Denver, Colo. 1904, Grad., West Side High School, Denver, Colo. 1904-05, Denver Univ. 1909, Grad., Colorado School of Mines, Met. Engr., 1909, Mill work, lead concentrator, Snowy Range Min. Co., Albion, Colo. 1909-10, Mill work, stamp mill; later assayer, Zopher Min. & Mill Co., Wallstreet, Colo. 1910-11, Asst. Supt. stamp mill, Tiger Gold Co., Harrington, Ariz. 1910, Instrumentman; 2d Asst. to Chief Engr., Eden Irrigation & Land Co., Eden, Wyo. 1911, Mech. Laboratory, Norton Co., Worcester, Mass.

Present position: Research Engr., Norton Co.

Robert LeRoy Hallett, Brooklyn, N. Y.

Proposed by Martin J. Heller, D. H. Browne, W. H. Aldridge.

Born 1881, Estes Park, Colo. 1905, Colorado School of Mines, E. M. 1905-08, Asst. Chem., Selby Smelt. & Lead Co., Selby, Cal. 1908-09, Chem., Von Schulz & Low, Denver, Colo. 1909-10, Cyanide Supt., Midas Gold Min. Co., Harrison Gulch, Cal. 1910-11, Chem. and Engr., Cons. Arizona Smelt. Co., Humboldt, Ariz.

Present position: 1911 to date; Min., Met., and Chem. Engr., National Lead Co.

Loyall Sherman Harner, Colorado Springs, Colo.

Proposed by A. L. Blomfield, F. W. Traphagen, J. F. Mitchell-Roberts.

Born 1872, Kansas. 1892, Kansas State College, B. Sc. 1899-1900, Mountain Copper Co., Keswick, Cal. 1901-02, U. S. Red. and Ref. Co. 1903-04, Portland Gold Min. Co. 1905-06, Mining in Nevada and Cal.

Present position: 1907 to date; Supt., Golden Cycle Min. Co.

Fred Negley Hays, Sharon, Pa.

Proposed by W. R. Crane, George V. Luerksen, C. E. McQuigg, Albert E. Rhoads, Robert M. Hutchison, H. D. Pallister.

Born 1891, Greensburgh, Pa. 1910, Pittsburgh Central High School. 1914, School of Mines, Pennsylvania State College, B. S. 1911-14, Furlough service as Chainman, Pennsylvania R. R. 1914-15, Instructor in Robert College, Constantinople, Turkey.

Present position: Asst. Steam Engr., Carnegie Steel Co., Farrell Works.

George Leston Helmrich, Rancagua, Chile, So. Amer.

Proposed by Ambrose E. Ring, George W. Roddewig, Robert Peele.

Born 1884, New Rochelle, N. Y. 1901-04, New Rochelle High School. 1904-09, Grad., School of Mines, Columbia Univ. 1909, Underground and cyanide plant, Standard Min. Co., Bodie, Cal. 1910-11, Foreman Engrg. Dept., Cyanide Plant, Honduras Rosario Min. Co., San Juancito, Honduras, New York. 1911-13, Engrg. Dept. and Shift Boss (underground), Ray Cons. Copper Co., Ray, Ariz.

Present position: 1913 to date; Genl. Mine Foreman, Braden Copper Co.

H. C. Henrie, Bisbee, Ariz.

Proposed by Arthur Notman, Charles A. Mitke, Roger T. Pelton Co.

Born 1885, Shamokin, Pa. 1902, High School. 1904, State Normal School, Bloomsburg, Pa. 1908, Penn. State College. 1904-05, summer, Reimard Bros., Bloomsburg, Pa. 1906-08, summer, Buck Ridge Coal Co., Shamokin, Pa.

Present position: 1908 to date; Chief Chemist, Copper Queen Cons. Min. Co.

Effingham Perot Humphrey, Drifton, Pa.

Proposed by Paul Sterling, R. V. Norris, J. M. Humphrey.

Born 1893, Wilkes-Barre, Pa. 1915, Lehigh Univ., M. E. 1911, summer, Machine Shop, Goynne Steam Pump Wks., Ashland, Pa. 1912, summer, Draughtsman, M. E. Dept., L. V. C. Co., Wilkes-Barre, Pa. 1913, summer, Inspector of construction, Franklin Bkr., L. V. C. Co., Wilkes-Barre, Pa. 1914, summer, Inspector of construction, Packer No. 5 Bkr., L. V. C. Co., Girardville, Pa.

Present position: 1915 to date; Engr. construction, Drifton No. 2 Bkr.

Frank E. Johnson, Salt Lake City, Utah.

Proposed by Ernest Gayford, C. W. Whitley, R. C. Gemmell.

Born 1877, Collegeville, Pa. 1884-94, Public Schools, Collegeville, Pa. 1894-97, Williamson Trade School, Williamson, Pa. 1897-99, Draftsman, American Bridge Co., Philadelphia. 1899-1901, Draftsman, Denver Engrg. Wks. Co.; Colo. Iron Wks. Co.; Stearns-Roger Mfg. Co., Denver. 1901-02, Draftsman and Assayer, Copper Queen Min. Co., Pictou, N. S. 1902-06, Associated with J. M. Callow and General Engrg. Co., Salt Lake City. 1906-09, Fairbanks-Morse & Co., Salt Lake City.

Present position: 1909 to date; Sales Engr., Amer. Manganese Steel Co.

J. Claude Jones, Reno, Nev.

Proposed by E. A. Julian, F. C. Lincoln, J. O. Greenan.

Born 1877, Merrimac, Wis. 1902, Univ. of Illinois, A. B. 1906-09, Grad. Student, Univ. of Chicago, Dept. of Geol. 1904-06, Instructor, Dept. of Geology, Univ. of Illinois. 1906-09, Field Asst., Illinois State Geological Survey. 1908-09, Research Asst., Dept. of Geology, Univ. of Chicago. 1909-12, Asst. Prof. Geology, Univ. of Nev.

Present position: 1912 to date; Prof. of Geol., Univ. of Nev.

Ralph W. Kerns, Warren, Ariz.

Proposed by Leon Feuchère, Arthur Notman, A. G. McGregor.

Born 1881, Tracy, Minn. 1900-03, Mankato Normal School, Minn. 1904-07, Univ. of Minn., E. E. 1907-08, Switchboard Attendant, Helena Power Trans. Co., Anaconda, Mont. 1909-10, Draftsman, International Smelt. and Ref. Co. 1910, Draftsman, Anaconda Copper Min. Co. 1911-13, Draftsman, Calumet and Arizona Min. Co. 1914-15, Mechanical Design, International Smelter, Miami.

Present position: 1915 to date; Chief Draftsman, New Cornelia Copper Co.

Sidney Albert Lang, Rancagua, Chile, Co. Amer.

Proposed by John P. Chadwick, W. S. Bishop, W. J. Turner.

Born 1884, Omemee, Ont., Canada. 1914, Grad., Univ. of Toronto, B. S. 1912-13, Mine Engrg. Dept., Canadian Copper Co., Copper Cliff, Canada.

Present position: Sampling Dept., Braden Copper Co.

William B. Lewis, New York, N. Y.

Proposed by F. W. Traphagen, William R. Chedsey, W. G. Haldane.

Born 1869, Bay City, Mich. 1892, Grad., Colorado School of Mines, M. E. & E. M., Chemist, Globe Smelter, Amer. Smelt. & Ref. Co., Denver. 1892-93, Genl. Mgr., Deming Ore Co., Deming, N. Mex., Engr. Santa Fe R. R. Co., N. Mex. Div. 1893-1900, Genl. Mgr., Denver Sulphite Fibre Co., Denver Mining interests, Leadville, Cripple Creek. 1900-09, Genl. Mgr., South Canon Coal Co., Denver, Trustee, Colo. School of Mines, Golden, Colo., Genl. Mgr., Western Sulphur Co., Idaho.

Present position: 1909-16, Prest., Oakdale Oil Co., New York City, Treas., English Oil Co., Tampico, Mex.

John Alexander McDonald, Campo Seco, Cal.

Proposed by H. S. Jordan, E. A. Hersam, A. P. Busey, Jr., W. S. Morley.

Born 1893, Benicia, Cal. 1907-11, Berkeley High School, Berkeley, Cal. 1911-15, Univ. of Cal., B. S. 1913, Miner, North Star Mines, Grass Valley, Cal. 1914, Miner, Harvard Mine, Jamestown, Cal. and Eagle-Shawmut Mine, Shawmut, Cal.

Present position: Chemist and Assayer, Penn Mining Co.

Alexander John McNab, Thompson, Lyon Co., Nev.

Proposed by John G. Kirchen, John L. Dynan, M. J. Conover.

Born 1877, Glergarry Co., Ont., Canada. Public and High Schools, Ont. 1902, Queens Univ., Kingston, Ont., Canada, B. A. 1902, School of Mining, Kingston, Ont., Canada, B. Sc. 1903-07, Met. and Research Chem. 1907-11, Met. and Smelter Supt., Cons. Min. & Smelt. Co., Trail, B. C.

Present position: 1912 to date; Asst. Mgr., Mason Valley Mines Co.

Arthur J. McQuatters, El Paso, Texas.

Proposed by Walter H. Aldridge, Donald C. Brown, David H. Browne.

Born 1874, Waxahachie, Texas. 1882-90, Public Schools of Texas. 1890. Private instructions along lines of civil and hydraulic engineering. 1900-09, Engaged in contracting business having constructed during this period more than one hundred public works, consisting of water works, drainage systems and electric plants.

Present position: 1909 to date, engaged in mining business. Now Prest. and Genl. Mgr., Alvarado Mining & Milling Co.

Frank Sanderson MacGregor, Boston, Mass.

Proposed by Henry A. Wentworth, Augustus H. Eustis, Charles H. White.

Born 1885, Lawrence, Mass. 1899, Hyde Park Grammar Schools. 1903, Hyde Park High School. 1907, Mass. Institute of Technology, S. B. 1907-12, Huff Electrostatic Separator Co.; United States Smelting Co.; American Zinc Lead & Smelt. Co.

Present position: 1912 to date; Asst. Treas., Met., Mount Champion Min. Co. and with H. A. Wentworth.

Walter Byron Miller, Harlan, Ky.

Proposed by Henry Groos, Will Ward Duffield, Alexander H. Wood.

Born 1878, Charleston, W. Va. 1892, Grad., High School, Charleston, W. Va. 1893-1904, W. Va. Univ., Morgantown, W. Va. 1895-98, General Practice, Min. Engr. 1899-1901, Univ. of Ky., Lexington, Ky. 1901-03, Genl. Mgr., Chapmans Coal & Coke Co., W. Va. 1903-04, Elec. Engr. 1904-13, Salesman and Elec. Engr., Morgan-Gardner Electric Co., Chicago, Ill.

Present position: 1914 to date; Genl. Supt., Harlan Gas Coal Co.

William Edgar Milligan, Rancagua, Chile, So. Amer.

Proposed by John P. Chadwick, W. S. Bishop, W. J. Turner.

Born 1891, Toronto, Canada. 1914, Grad., Univ. of Toronto, B. S. 1914-15, Assaying and Sampling Depts., Braden Copper Co.

Present position: Assaying Dept., Braden Copper Co.

Luke Mooney, Bayonne, N. J.

Proposed by R. M. Atwater, Jr., George E. Farish, Felix A. Vogel.

Born 1869, Kingston, N. Y. With the Standard Oil Co., Bayonne, N. J., in the following capacities. 1882-92, Boy Laborer, Clerk, Asst. Chemist, Supt. of Glue Factory. 1892-1902, Supt. of Dept. for preparing, filling and shipping oil in bulk and packages. 1902-04, Special work with fuel used under boilers and stills to improve efficiency and economy. 1904-11, Supt. of Paraffine Oil and Wax Refinery. With H. Clay Pierce. 1911, Erecting wax plant, Tampico, Mexico, organizing operating force, etc. 1912, Constr. Oil Refinery, Fort Worth, Tex., organizing force; and later Supt. in charge of operation. 1913-14, Genl. Supt., Oil Refineries, Ft. Worth, Tex., Texas City, Tex. and Tulsa, Okla. 1915, Expert work in manufacture of wax, The Texas Co., Port Arthur Refinery, Port Arthur, Tex. 1915, Expert work in manufacture of wax, The Gulf Ref. Co., Port Arthur Refinery, Port Arthur, Tex. 1915, Expert work improving efficiency and economy of power plants, J. H. & C. K. Eagle Silk Mfrs., N. Y., Shamokin, Pa.

Present position: President, Constructing New Jersey Glue Co. Plant, Perth Amboy, N. J.

John Carroll Moulton, Blair, Nev.

Proposed by Lyon Smith, C. W. Merrill, L. D. Mills.

Born 1876, Laconia, N. H. 1894-96, Dartmouth College. 1896-97, Stanford Univ. 1904-07, Tucabe G. M. Co., Magdalena, Son., Mex. 1907-10, Pittsburg Silver-Peak G. M. Co., Blair, Nev. 1911-15, Merrill Metallurgical Co., San Francisco, Cal.

Present position: Mill Supt., Pittsburg Silver-Peak G. M. Co.

Mortimer S. North, Calumet, Mich.

Proposed by James MacNaughton, F. W. McNair, F. W. Sperr.

Born 1890, Calumet, Mich. 1909-11, Dept. of Civ. Engrg., Univ. of Michigan. 1912-15, Michigan College of Mines, Houghton, Mich., E. M., B. Sc. 1910, summer, 1911, winter, Civil and Min. Engrg. Dept., Calumet & Hecla Min. Co., Calumet, Mich.

Present position: 1915 to date; Electrolytic Plant, Calumet & Hecla Min. Co.

Pablo Ortega, Havana, Cuba.

Proposed by E. Aguilera, Pedro Aguilera, Eduardo J. Chibas.

Born 1865, Havana, Cuba. 1888, Grad., Ecole Polytechnique de Bruxelles (Belgium), Min. Engr. 1889-1906, Private Engrg. Practice and R. R. Surveying. 1896-1901, In the U. S., Krajewski & Pesant, and R. Marsans & Co. 1901-09, Chief Engr., Public Works Dept.

Present position: 1909 to date; Director, Mines and Forestry Dept.

Calvin Pardee, Jr., Hazleton, Pa.

Proposed by H. M. Crankshaw, J. Elmer Jones, Edwin Ludlow.

Born 1871, Hazleton, Pa. 1889-91, Princeton. 1892-94, Asst. Supt., Pardee Works. 1894-1907, Asst. Supt. 1907-11, Vice-Pres., Pardee Bros. & Co., Inc.

Present position: 1911 to date; Prest. and Genl. Mgr., Pardee Bros. Co., Inc.

Isak Partanen, Telluride, Colo.

Proposed by James S. James, Charles N. Bell, Walter L. Reid.

Born 1887, Utajarvi, Finland. High School, Lahti, Finland. 1907, Passed entrance examinations at Univ. of Helsingford. 1909-12, Michigan College of Mines, E. M. 1912, Inspector of construction, Munro Iron Min. Co., Iron River, Mich. Practical mining, etc., with the purpose of studying methods and conditions in different mining camps of Michigan, Utah, Montana and Idaho. 1914, Min. Engr., Idaho-Continental Min. Co., Port Hill, Ida.

Present position: Min. Engr., Colorado Superior Min. Co.

James S. Pates, Irwin, Pa.

Proposed by S. A. Taylor, W. W. Keefer, S. A. Scott.

Born 1872, Midway, Ky. Common school. 1888-89, Brakeman, L. & N. R. R. 1892, Condr., Kentucky Midland R. R. 1900, Tobacco trade. 1906, Engrg. Dept., M. R. C. C. & C. Co. 1907, Engrg. Dept., Monongahela Mfg. Co. 1915, Mgr., mining coal.

Present position: Prest., Export Coal Co.

Clarence J. Peterson, Tonopah, Nev.

Proposed by W. H. Blackburn, Frederick Bradshaw, J. E. Spurr.

Born 1885, Honolulu, T. H. 1902, Grad., Honolulu High School, Honolulu. 1910, Leland Stanford Jr. Univ., A. B. 1902-05, Asst. Cashier, Hind, Ralph & Co., Honolulu. 1910-12, Asst. Engr., Tonopah Min. Co., Tonopah, Nev. 1912, Field Geol., The Bermudy Co., Petroleum investigations in Districts of Benitez, Arismende, Marino, State of Lucre, Venezuela.

Present position: 1912 to date; Min. Engr., Tonopah Min. Co.

Charles Edward Prior, Hedley, B. C., Canada.

Proposed by M. K. Rodgers, W. G. Haldane, F. W. Traphagen.

Born 1891, Cincinnati, O. 1913, Grad., Colorado School of Mines, E. M. 1914, Post-graduate, Columbia School of Mines, A. M. 1913, Supt., Diamond Drilling Exploration Syndicate, No. 2, Hedley, B. C. 1914, Engr., Hedley Gold Min. Co.

Present position: 1914 to date; Assayer, Hedley Gold Min. Co.

Palmer Chamberlaine Ricketts, Troy, N. Y.

Proposed by L. D. Ricketts, R. W. Raymond, W. G. Sharp, R. W. Hunt, C. E. Mills.

Born 1856, Elkton, Md. 1875, Rensselaer Polytechnic Institute, C. E. 1905, Stevens Institute, E. D. 1911, New York University, LL. D. 1875-82, Instructor; 1882-84, Asst. Prof. 1985, Prof. Technical Mechanics. 1892-1901, Director. 1901-16 President and Director, Rensselaer Polytechnic Institute. President, Lakeview Sanatorium. Director, United National Bank. Trustee, Albany Academy. Director, Samaritan Hospital. Cons. Engr., railway and other corporations. Member, Amer. Soc. C. E. (director 1899-1901; vice-president 1916): Amer. Soc. Mech. Engr., Instn. C. E., Great Britain. Clubs: Troy (Troy), Union (New York). Author: History of Rensselaer Polytechnic Institute, 1895, revised 1915. Contributor to technical journals and transactions.

Present position: Prest., Rensselaer Polytechnic Institute.

Rudolph R. Rosenbaum, Chicago, Ill.

Proposed by Henry W. Nichols, J. A. Ede, H. B. Pulsifer.

Born 1884, Mühr, Ostrau. 1900, Gymnasium, Mühr, Ostrau. 1904-06, Imperial College, Birlitz Univ., Berlin. Dr. Behiend, Hamburg (sworn custom house chemist). Oderfurter Mineral oil refineries. A. G. first Chemist, later Supt., Mühr, Ostrau. Arcia Levick Co., Philadelphia. Atlantic Ref. Co., Philadelphia.

Present position: Chemist, Central Commercial Co.; Wabash Ref. Co.; Keystone Oil and Mfg. Co.

John Moore Samuel, Douglas, Ariz.

Proposed by Forest Rutherford, Hylton H. Colley, H. O. Hammond.

Born 1888, Wrexham, No. Wales. 1898-1906, Grove Park School, Wrexham. 1906-09, Grad., Royal School of Mines, London, England, Dept. of Met. 1909-11, Post Grad. course, Copper Queen Smelter.

Present position: 1911 to date; Testing Engr., Copper Queen Smelter.

David Burnet Scott, Miami, Ariz.

Proposed by B. Britton Gottsberger, F. W. MacLennan, H. P. Bowen.

Born 1881, Slatington, Pa. 1904-08, Williams College, A. B. 1908-11, Columbia Univ., E. M. 1909, Civil Engr., John Farley Co., Westchester Co., N. Y. 1911, Leasing and prospecting, Colorado.

Present position: 1911 to date; Efficiency Engr., Miami Copper Co.

James Graves Scrugham, Reno, Nev.

Proposed by E. A. Julian, F. C. Lincoln, J. O. Greenan.

Born 1880, Lexington, Ky. 1896-1900, State Univ. of Ky., B. M. E. 1906, M. E. 1899-1905, Technical positions with C. N. O. & T. P. Ry.; Creaghead Engrg. Co.; Metropolitan Elevated Ry. Co.; Cal. Elec. Wks. Abner Doble Co.; Univ. of Nev., etc. 1905-14, Prof. of Mech. Engrg., Univ. of Nev. Consulting engineer on plans, tests, specifications, etc., for Oregon Short Line Ry.; Southern Pac. Co.; Buckhorn Mines Co.; Nev.-Cal.-Oregon Ry.; Elko Power Co.; Tr. Riv. Gen. Elec. Co.; Reno Power, Light and Water Co.; Reno Traction Co.; Rochester Hills Min. Co.; Nev. Short Line Ry.; Public Utilities Commission, etc.

Present position: 1914 to date; Dean of Engrg., Univ. of Nevada.

Luther Crocker Snider, Norman, Okla.

Proposed by E. DeGolyer, Albert D. Brokaw, H. A. Buehler.

Born 1882, Mt. Summit, Ind. 1903-04, Rose Polytechnic Institute. 1905-09, Indiana Univ., A. B., A. M. 1914-15, Univ. of Chicago, Ph. D. 1909-10, Asst. to State Geologist of Indiana. 1910-15, Chem. Asst. Director and field Geol., Oklahoma Geological Survey.

Present position: Geol., Pierce Oil Corp.

Theron D. Stay, Cleveland, O.

Proposed by Charles H. Fulton, Zay Jeffries, J. Burns Read.

Born 1889, Yankton, S. D. 1903-07, Lincoln High School, Cleveland, O. 1910-14, Case School of Applied Science, B. S. 1907-10, Wells Fargo & Co. Express, Cleveland, O. 1914-15, Met., National Transit Pump and Mach. Co., Oil City, Pa.

Present position: Asst. Research Met., Aluminum Castings Co.

Warren Nelson Thayer, Cincinnati, O.

Proposed by H. M. Waite, John J. Porter, G. M. Lambourne.

Born 1883, Cincinnati, O. 1889-1901, Public and High Schools, Cincinnati. 1902-03, Normal College, Fenton, Mich. 1903-04, Miami Univ., Oxford, O. 1906-12, Special student in Science; 1912-15, post-graduate work in Geology, Univ. of Cincinnati. 1912-13, Geol. and Chem., Chas. Taylor Sons Firebrick Co., and the Michigan Limestone Products Co., Cincinnati. 1913-14, Asst. in Geol., Univ. of Cincinnati. 1915, Field season, examination of various zinc mine and smelter dumps in Ky. and Tenn.

Present position: 1913-15, Prof. of Geology, Ohio Mechanics Institute.

Paul McIntosh Tyler, Breckenridge, Colo.

Proposed by Charles E. Locke, H. O. Hofman, Carle R. Hayward.

Born 1889, Oberlin, O. 1912, Mass. Institute of Technology, S. B. 1912-13, Asst. in Chem., Mass. Institute of Technology. 1913-14, Engr., Moctezuma Copper Co., Pilares de Nacozari, Son., Mex. 1914-15, Experimental work, Boston, Mass.

Present position: Supt., The Rilla Min. Co.

Thomas Varley, Flat River, Mo.

Proposed by Ambrose E. Ring, George W. Roddewig, C. J. Adami.

Born 1877, Salt Lake City, Utah. 1893-96, High School, Salt Lake City, Utah. 1903-07, School of Mines, Univ. of Utah, B. S. 1898-1903, Millman and repairman, mill, Daly-West Min. Co., Park City, Utah. 1907-08, Mill Supt., Atlanta Mines Co., Atlanta, Idaho. 1908-09, Transitman, surveying party, southern Utah. 1909-12, Leasing and operating small cyanide plant, Rye Patch, Humboldt Co., Nev.

Present position: 1912 to date; Ore testing dept., Federal Lead Co.

H. J. Wasson, Rockland, Nev.

Proposed by Curtis F. Burt, M. F. Quinn, George J. Young.

Born 1890, Laporte City, Ia. 1907, Morgan Park Academy. 1914, Univ. of Minnesota, E. M. 1910-11, Miner, Copper Queen Copper Min. Co., Bisbee, Ariz. 1913, Miner, Anaconda Copper Min. Co., Butte, Mont.

Present position: 1914 to date; Min. Engr., Pittsburg-Dolores Min. Co.

W. A. I. M. van Waterschoot van der Gracht, Tulsa, Okla.

Proposed by W. E. Wrather, Charles H. Taylor, E. T. Dumble.

Born 1873, Amsterdam, Holland. 1899, Univ. of Amsterdam, LL. D., 1903, Certificated Min. Engr. and Geol., Freiberg, Saxony. 1903, Member, Government Min. Council, Netherlands. 1905, Director, Netherlands Geological Service (now on leave). Geol. and Min. Engrg. experience in British Columbia, South Africa, Magellan Territories, Dutch East Indies, Roumania, Russia, Spain, Portugal, Germany, Belgium, England and the Netherlands, in employ either of the Government or of various syndicates or companies, notably the Orion Petrol. Co., and the Royal Dutch Shell Combine.

Present position: Advisor to the Roxana Petroleum Co., and the Royal Dutch Shell Combine.

Albert Wauchope, Gwalia, West. Aust.

Proposed by G. C. Klug, J. W. Sutherland, R. A. Varden.

Born 1870, Kapunda, South Aust. 1889-94, Assaying, Chemistry, Metallurgy and kindred subjects, South Australian School of Mines and Adelaide Univ. 1894-96, Assayer, Met., Virginia and New Milo Gold Mines, Wadnaminga, S. Aust. 1896-97, Managing, Cobar Development Syndicate, N. S. W. 1897-1900, Managing, Woodside Development Co., S. Aust. 1900-05, Met., Gt. Boulder Main Reef and Fremantle Smelt. Works; operating mine at Menzies, W. A. 1905-10, Genl. Mgr., Gt. Boulder Main Reef, Boulder, W. A.

Present position: 1910 to date; Supt., Sons of Gwalia, Ltd.

Henry S. Winans, Denver, Colo.

Proposed by Ernest Gayford, C. W. Whitley, R. C. Gemmell.

Born 1857, La Fayette, Ind. Common School. No technical education except what I have absorbed by reason of my business around mines and mills for about 25 years. 1884-89, Bookkeeper, Special Agent, Mgr., The Standard Oil Co. 1890-99, Western Representative, Wadhams Oil Co.

Present position: 1899 to date; Representing The W. S. Tyler Co.

Associate Member

George Graham Titzell, Jr., Duquesne, Pa.

Proposed by A. N. Diehl, J. S. Unger, C. F. W. Rys.

Born 1892, Kittanning, Pa. 1906, Kittanning Public Schools. 1910, Kittanning High School. 1914, Sheffield Scientific School, Ph. B. 8 months, D. C. Morgan, Genl. Supt., C. E. Dept., Pittsburgh and Shawmut R. R.; 4 months, William Affeldar, Genl. Mgr., Engrg. Dept., Bessemer Coke Co., Pittsburgh, Pa., 6 months, Franke B. Schaeffer, Chief Engr., Engrg. Dept., Homestead Steel Wks., Munhall, Pa.

Present position: Student, Blast Furnace Dept., Duquesne Steel Wks.

Junior Members

George K. Brennen, Scottdale, Pa.

Proposed by W. R. Crane, W. M. Weigel, C. E. McQuigg.

Born 1893, Scottdale, Pa. 1899-1912, Scottdale Public and High School. 1913, summer, Engrg. corps, Mr. John M. Rayburn, Pittsburgh. 1915, summer, School of Mines Surveying trip, Bituminous region, Pa.; also one month, timberman, Thompson No. 1 Mine, Thompson-Connellsville Coke Co.

Present position: Senior, School of Mines, Penn. State College.

Robert Stanley Burg, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born ———, 1901-04, Tempe Normal School, Tempe, Ariz. 1908, Rodman and Chairman, in a railroad, Mammoth, Ariz., mine and topog. survey, Calumet and Arizona Min. Co., Bisbee, Ariz. 1909, Asst. Engr., U. S. Deputy Mineral Survey, Clifton, Ariz. 1909-11, Millman and Surveyor, Arizona Copper Co., Morenci, Ariz. 1911-13, Chief Mine Surveyor, Arizona Copper Co.

Present position: Student, Missouri School of Mines.

John Darrah Cooner, Watsontown, Pa.

Proposed by W. R. Crane, C. E. McQuigg, W. M. Weigel.

Born 1895, Watsontown, Pa. 1902-08, Watsontown Public Schools. 1909-12, Watsontown High School.

Present position: 1912 to date; Student, Penn. State College.

Harry Gilbert Corby, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1895, Pleasanton, Kans. 1913, Grad., Carthage High School, Carthage, Mo. 1912, 1913-14, Frank B. Newton, Min. and Civ. Engr., Carthage, Mo. 1915, Drill Helper and Drillman, American Zinc, Lead & Smelt. Co., Carterville, Mo.

Present position: Student, Missouri School of Mines.

F. Nicholas Donero, Reno, Nev.

Proposed by F. C. Lincoln, E. A. Julian, J. O. Greenan.

Born 1890, Reno, Nev. 1912, summer, Nevada Hills Min. Co., Fairview, Nev. 1913, summer, Nevada Hills Min. Co., Fairview, Nev. 1914, summer, A. Dondero Min. Co., Olinghouse, Nev. 1915, summer, Buckhorn Mines Co., Buckhorn, Nev.

Present position: 1911 to date; Student, Mackay School of Mines.

Louis Wilmer Ehlers, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1893, Baltimore, Md. Baltimore Public Schools. 1910-12, Baltimore Polytechnic Institute. 1912, summer Harvard Engineering Camp. 1914, Underground, Cobalt Tourisite Min. Co., Cobalt, Ont. 1914, Dominion Red Co., Cobalt, Ont. 1915, summer, underground, American Lead, Zinc and Smelting Co., Carterville, Mo.

Present position: Student, Missouri School of Mines.

E. P. Hammitt, Apollo, Pa.

Proposed by W. R. Crane, C. E. McQuigg, W. M. Weigel.

Born 1893, Apollo, Pa. 1899-10, Apollo Public School and High School. 1910-12, Mercersburg Academy. 1913, summer, three months, Mech. Engrg. Dept., Amer. Sheet & Tin Plate Co., Vandergrift, Pa. 1914, summer, two months, Experiment Engr., Amer. Sheet & Tin Plate Co., Vandergrift Works. 1915, summer, three week summer school trip in the soft coal region of Pa.

Present position: Student, Penn. State College.

Max Travis Hofius, Golden, Colo.

Proposed by William R. Chedsey, F. W. Traphagen, W. G. Haldane.

Born 1895, Belize, Brit. Honduras. 1915, summer, American Oil Flotation Co., Silverton, Co. and Sunnyside Mines, Eureka, Colo.

Present position: Student, Colorado School of Mines.

Earl A. Jones, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1894, Rolla, Mo. 1908-12, Grad., Rolla High School. 1914, summer employed in mill and refinery, Buffalo Mines, Ltd., Cobalt, Ont., Can. 1915, summer employed in smelter and laboratory, National Zinc Co., Bartlesville, Okla.

Present position: Student, Missouri School of Mines.

Thomas Harold Kernan, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1891, Newaygo, Mich. 1910-11, Univ. of Wisconsin. 1911-12, Illinois Steel Co., Gary Works.

Present position: 1913 to date; Student, Univ. of Wisconsin.

Harry Albert Kluge, Rolla, Mo.

Proposed by C. R. Forbes, G. H. Cox, D. H. Radcliffe.

Born 1895, Collinsville, Ill. 1909-13, Collinsville Township High School. 1914, Labor, St. Louis Smelt. & Ref. Co., Collinsville, Ill. 1915, Labor, Portland Min. & Mill. Co., Victor, Colo.

Present position: Student, Missouri School of Mines.

Hugo Edward Koch, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1891, Creve Coeur, Mo. 1906-10, Manual Training School, St. Louis, Mo. 1912, summer, County Engineers Office, Clayton, Mo.

Present position: Student, Missouri School of Mines.

Norman Eyre Maxwell, Golden, Colo.

Proposed by William R. Chedsey, F. W. Traphagen, W. G. Haldane.

Born 1894, Silverton, Colo. Practical experience in the mines and mills in San Juan District, Colo.

Present position: Student, Colorado School of Mines.

Thaddeus Roderick Noon, Peru, Ill.

Proposed by J. A. Ede, Henry W. Nichols, H. A. Buehler.

Born 1893, Peru, Ill. 1908-13, St. Marys College, St. Marys, Kans. During vacations worked for Mr. Ede, Min. Engr., Illinois Zinc Co.

Present position: 1913-15; Marquette Univ., Milwaukee, Wis.

Harold Lippert Rau, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1889, Milwaukee, Wis. 1908-10, Univ. of Wisconsin. 1907-08, Oliver Iron Min. Co., Coleraine and Marble. 1910-11, Engrg. Dept., Chicago and North Western R. R. Co., Iowa and I. & M. Div. 1911-12, Engrg. Dept., Union Oil Co. of Cal. 1912-13, in Mill, Pittsburgh Silver Peak Gold Min. Co. 1914, Blackhills-Homestake. 1915, Cleveland Min. Co., Hazel Green, Wis.

Present position: 1913 to date, Student, Univ. of Wisconsin.

Joseph Gaskill Skirm, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1889, Princeton, N. J. 1911, Grad., Phillips Exeter Academy. 1911-12, Univ. of Wisconsin. 1913, Underground Miner, Wisconsin Zinc Co., Benton, Wis. Present position: 1913 to date; Univ. of Wisconsin.

Earl B. Weiberg, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1895, Peoria, Ill. 1909-12, High School, Springfield, Ill. 1913, summer, National Zinc Co., Springfield, Ill. 1914, summer, sampling, National Zinc Co., Springfield, Ill. 1915, summer, Asst. Chem., National Zinc Co., Springfield, Ill. Present position: Student, Missouri School of Mines.

Daniel S. Whiteman, So. Bethlehem, Pa.

Proposed by Howard Eckfeldt, Joseph W. Richards, Benjamin L. Miller.

Born 1893, Philadelphia, Pa. 1908-10, Central High School, Philadelphia, Pa. 1910-12, Central Manual Training High School, Philadelphia, Pa. 1915, with Engineer of Construction, P. R. R., Philadelphia, Pa.

Present position: Secy., Reliance Coal & Coke Co., Hartranft, Claiborne Co., Pa. and Student, Lehigh Univ.

Harry Raymond Wilson, Golden, Colo.

Proposed by W. G. Haldane, F. W. Traphagen, Harry J. Wolf.

Born 1889, Denver, Colo. 1904-08, East Denver High School. 1909-12, Colorado School of Mines. 1911, Mechanic's Helper, Wellington Mines Co., Breckenridge, Colo. 1913-15, Chem., Von Schulz & Low, Denver.

Present position: Student, Colorado School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Jan. 10, 1916 to Feb. 10, 1916. This list, together with the list published in *Bulletin* No. 110, Feb. 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916, and brings it up to the date of Feb. 10, 1916.

ADAMS, A. K.	Box 100, Miami, Ariz.
ALEXANDER, W. H.	45 Broadway, New York, N. Y.
ALSDORF, F. C.	75 W. Willetta St., Phoenix, Ariz.
AMIDON, R. G., Mgr.	Queen of the West Mines Co., Cornucopia, Ore.
ANGERER, V.	Box 214, Easton, Pa.
ARMS, CHARLES S.	Midvale Steel Co., Philadelphia, Pa.
BARKER, H. A.	Care Sres. Stafford & Co., Arequipa, Peru, So. Am.
BARNES, W. A.	211 Luzerne Ave., Pittston, Pa.
BASSETT, THOMAS E.	Box 277, Bisbee, Ariz.
BATCHELLER, J. H.	Box 36, Mattapoisett, Mass.
BECKWITH, H. T., Chief Geol., Indian Territory Illuminating Oil Co.	Bartlesville, Okla.
BELL, J. MACKINTOSH, The Huronian Belt Co., Ltd.,	310 Dominion Express Bldg., Montreal, Can.
BETTERTON, JESSE O.	Amer. Smelt. & Ref. Co., Omaha, Nebr.
BEYER, SAMUEL W.	Iowa State College, Ames, Iowa.
BLAKE, D. E.	Kingman, Ariz.
BLOMFIELD, A. L.	810 Cooper Bldg., Denver, Colo.
BORIE, ADOLPHE E.	Room 1550, 50 Church St., New York, N. Y.
BRADFORD, ROBERT H.	University of Utah, Salt Lake City, Utah.
BREWER, A. K.	1404 Baker St., San Francisco, Cal.
BRITT, RICHARD H.	76 Virginia St., Springfield, Mass.
BULKLEY, RALPH G., Min. Engr.	Marion Mines & Mill Co., Denver, Colo.
BUTLER, M. C.	Waldorf Hotel, Seattle, Wash.
CARPENTER, JAMES R., Care F. N. Cameron,	811 Kearns Bldg., Salt Lake City, Utah.
CASE, ALBERT H.	Care General Development Co., 61 Broadway, N. Y.
CHALMERS, J. W. P.	Lynne Regis, Dorset, England.
CHASE, F. D.	Apartado 27, Monterrey, N. L., Mex.

- CHIBAS, LUIS F. Mgr. Cia de Electricidad, Guantanamo, Cuba.
 CHIDESTER, W. B. Luning, Nev.
 CHOW, KAI CHI. Wah Chang M. & S. Co., Changsha, Hunan, China.
 CLAPP, F. G., Managing Geol., The Associated Geological Engineers,
 Room 3112, 120 Broadway, New York, N. Y.
 CLAPP, L. R. 1137 Sherman St., Denver, Colo.
 CLARKE, SIMEON S., Supt. of Leadwood Division, St. Joseph Lead Co., Leadwood, Mo.
 COX, W. RAY. 310 Custom House, Portland, Ore.
 CRANKSHAW, H. M. Mgr., Harwood Coal Co., Hazleton, Pa.
 CUNNIFF, BERNARD. 53 State Street, Boston, Mass.
 CUNNINGHAM, WALTER H., 800-802 First National Bank Bldg., Huntington, W. Va.
 DESHLER, GEORGE O. Instructed to hold all mail.
 DEVEREUX, JAMES H. 120 Broadway, New York, N. Y.
 DEVEREUX, WALTER B. 120 Broadway, New York, N. Y.
 DINGWALL, W. B. A. 516 Hicks Bldg., Box 179, San Antonio, Tex.
 DOWNS, FLETCHER G. 142 Greene St., Hudson, N. Y.
 DUNN, T. S. Care Palmer House, Patton, Pa.
 EATON, A. L. Room 1206, Mill Bldg., El Paso, Tex.
 ELMER, WILLIAM W. 717 E. Broadway, Portland, Ore.
 EMLAW, H. S. Grand Haven, Mich.
 ENZIAN, CHARLES, Min. Engr. U. S. Bureau of Mines, Pittsburgh, Pa.
 EVANS, GEO. WATKIN. 901 American Bank Bldg., Seattle, Wash.
 FABIAN, FRANCIS G. 521 West Adams St., Chicago, Ill.
 FARISH, JOHN B. 58 Sutter St., San Francisco, Cal.
 FAWCETT, J. H. Care Bank of Australasia, Perth, Western Aust.
 FITZGERALD, J. M. 307 St. Paul St., Rochester, N. Y.
 FRASER, COLIN, Care The Broken Hill Associated Smelters Pty., Ltd.,
 Collins House, Melbourne, Vic., Aust.
 GANNON, MICHAEL H. 206 Pennsylvania Bldg., Butte, Mont.
 GARD, IRVING R. Instructed to hold all mail.
 GARREY, GEORGE H. 501 Bullitt Bldg., Philadelphia, Pa.
 GEHRES, GEORGE W. 27 East Ridge St., Lansford, Carbon Co., Pa.
 GLASS, FRANK A., Acting Supt., Wilcox Mine, Paterson Construction Co., R.F.D. 2,
 Brainerd, Minn.
 GLEESON, WALTER G. 1112 Northwestern Bank Bldg., Portland, Ore.
 GRANT, LESTER E. Care J. B. Grant, 770 Penn. St., Denver, Colo.
 HAMILTON, CHARLES W., ... Care Roma Oil Co., 202-208 Lynch Bldg., Tulsa, Okla.
 HANAHAN, M. L. Hartley Hall, Columbia University, New York, N. Y.
 HARBACH, HERBERT M. Lebanon, Pa.
 HART, C. R. Hermosillo, Son., Mex.
 HART, VERNE A. 620 Kearns Bldg., Salt Lake City, Utah.
 HAYDEN, RALPH, Supt. of Regrinding and Flotation, Anaconda Copper Min. Co.,
 Anaconda, Mont.
 HAYES, FRANCIS H. Box 416, Morenci, Ariz.
 HEIZER, OTT F. Box 687, Idaho Springs, Colo.
 HENRICH, CARL. Magruder Mine, Lincoln, Ga.
 HENRY J. T. Lehigh Coal and Navigation Co., Lansford, Pa.
 HOFFER, ALLEN, ... Supt. Blast Furnaces, Worth Brothers Co., Coatesville, Pa.
 HOLZWORTH, CHARLES R. West Middlesex, Pa.
 HOOKER, W., 75 Parliament Hill Mansions, Gospel Oak, Highgate,
 N., London, England.
 HOOPER, E. 5 London Wall Bldgs., London, E. C., England.
 HOTCHKIN, MERRITT W., Mgr., Tough-Oakes Gold Mines, Ltd.,
 Kirkland Lake, Ont., Can.
 HOWELL, FRANKLIN DAVENPORT, Chief Engr., Department of Public Utilities,
 Room 23, City Hall, Los Angeles, Cal.
 HUMPHREY, GEORGE S., Care C. W. Hunt. & Co., Inc., 61 Broadway,
 New York, N. Y.
 HUNTER, HANSABURO, Care E. H. Hunter & Co., 14 Honden, Nibancho Nishiku,
 Osaka, Japan.
 HYDE, JAMES M. 634 Mills Bldg., San Francisco, Cal.
 ISHIHARA, KIUKICHI, The Tsunatori Mine, Yokokawame-Mura, Waka-Gun,
 Iwate-Ken, Japan.
 JAMES, W. EWART, ... Genl. Supt., Mount Carbon Co., Ltd., Powellton, W. Va.
 JEFFS, LEWIS A. Jeffs & Johnson, 406 Dooly Bldg., Salt Lake City, Utah.
 JENSEN, JOSEPH. 512 Custom House, San Francisco, Cal.

- JOHNSON, F. A. Care Standard Oil Co., Shanghai, China.
 JOHNSON, J. FREDERICK ... Mgr., California Chief Development Co., Forest Hill, Cal.
 JOHNSON, W. MCA. Care Kusa Spelter Co., Dewar, Okla.
 JOSEPHS, IRVING S. 450 Riverside Drive, New York, N. Y.
 JUDSON, WILBUR 714 Mills Bldg., El Paso, Tex.
 KAANTA, HENRY W. Telluride, Colo.
 KASSON, BURT Z. 70 Fremont St., Gloversville, N. Y.
 KIRKPATRICK, S. F. Kensington Avenue, Kingston, Ont., Canada.
 KURLA, M. H., Vice-Prest. and Genl. Mgr., Care Broadwater Mills Co.,
 Park City, Utah.
 LAUER, HERBERT H. 228 W. Seymour St., Germantown, Philadelphia, Pa.
 LAURIE, FRANK C., Care Pacific Pipe Line Co., Title Insurance Bldg.,
 Los Angeles, Cal.
 LAWTON, N. O., Mgr. Explorations. Sneedville, Hancock Co., Tenn.
 LEE, HOWARD S. Box 948, Leadville, Colo.
 LELAND, E. Benson, Ariz.
 LINDEMUTH, L. B. Steelton, Pa.
 LIST, ELMER 901 Carolina St., Neodesha, Kans.
 LIVINGSTON, IVOR First Creek Coal Co., Hazard, Ky.
 LONDON, C. J. 1528 Master St., Philadelphia, Pa.
 LOWER, J. B., Supt. Tabowie and Taracot, Oriental Cons. Min. Co.,
 Unsan Kinko, Chosen, Korea, Asia.
 LYMAN, FRANK, Mgr. 14 Wall St., New York, N. Y.
 MCCURE, DAVID University Club, San Francisco, Cal.
 MCKIM, JOHN W. 280 East 4th So. St., Salt Lake City, Utah.
 MACARTHUR, JOHN S., Lock Lomond Radium Wks., Balloch, Dumbartonshire,
 Scotland.
 MACAULAY, R. M. Grande Allee St., Quebec City, Canada.
 MANAHAN, ROBERT F. 1112 Mills Bldg., El Paso, Tex.
 MANN, WILLIAM SEWARD, Min. and Met. Engr.; Mill Supt., Calumet and Sonora
 of Cananea Mining Co., Cananea, Sonora, Mexico.
 MANNING, VAN H. 3602 Newark St., Washington, D. C.
 MARSH, HARRY W. Care Federal Mining and Smelting Co., Wallace, Ida.
 MILES, JOHN B. Ardmore, Pa.
 MILLER, HARRY H., Mgr., Cia Minera Lo Increible, El Callas, Yuruary, Venezuela,
 So. A., via Trinidad.
 MISHLER, RALPH T. Care The Tigre Mining Co., Douglas, Ariz.
 MOORE, CHARLES J. Room 801 Equitable Bldg., Denver, Colo.
 MOORE, REDICK R. Supt., Beaver Creek Mines, Zortman, Mont.
 MONTOLIEU, E. I. United States Mint, Philadelphia, Pa.
 MORE, GEORGE A. Murarrie Lindfield, Sydney, N. S. W., Aust.
 MORGAN, HARRY J. Box 1054, Stanford University, Cal.
 MUIR, DOWNIE D., JR., Mgr., Intermountain Mine, U. S. Smelt., Refg., & Min.
 Exploration Co., Salt Lake City, Utah.
 NAGEL, F. J., Care Cia. Minera de Penoles, S. A., 1008 Mills Bldg., El Paso, Tex.
 NEWELL, ARTHUR, The Flood Block, Anaconda, Mont.
 OFFICER, H. G. 529 W. 111th St., New York, N. Y.
 ORMSBEE, JAMES J. Box 681, El Paso, Tex.
 PAGE, J. M. Tyrone, N. M.
 PATTERSON, S. B., JR., Supt., The Spanish-American Iron Co., Daiquiri,
 Santiago de Cuba, Cuba.
 PERRY, EDWARD H., Min. Geol. Geological Museum, Cambridge, Mass.
 PHILPOTT, ROYDEN, C. Box 445, Cripple Creek, Colo.
 PITMAN, S. M. Box 1118, Providence, R. I.
 POSTLETHWAITE, R. H., Mgr., Menakher Date Garden, Coachella, Riverside Co., Cal.
 POWER, HAROLD T., Chief Engr., State Water Commission of Cal., 632 Call Bldg.,
 74 New Montgomery St., San Francisco, Cal.
 PRICKETT, W. C. 1145 North 12th St., Birmingham, Ala.
 PROUT, JOHN W., JR. Supt., Mascot Copper Co., Dos Cabezas, Ariz.
 RALSTON, W. C. 25 Broad St., New York, N. Y.
 RAY, JAMES C. Oatman, Ariz.
 RAYMOND, R. M. School of Mines, Columbia Univ., New York, N. Y.
 RETTIE, WILLIAM H., Care Forminiére Mission, Kasai, Tshikapa, Congo Belge, Africa.
 RICE, GEORGE S. Care Bureau of Mines, Washington, D. C.
 RICHMOND, WILLIAM H. Varadena Beach, Cardenas, Cuba.
 RINGLUND, S. Socorro, N. Mex.

- RIORDAN, D. M.....525 Market St., San Francisco, Cal.
 RODGERS, SELDEN S.....400 W. 4th St., Anaconda, Mont.
 SALISBURY, E. F., Care Cia. Minerales y Metales, S. A.,
 Apartado 251, Monterey, Mex.
 SAYRE, EDWARD A.....201 7th Street, Des Moines, Ia.
 SCHLERETH, C. Q., Cia. Minerales y Metales, Apartado 251, Monterey, N. L., Mexico.
 SCHNEPP, CHARLES F.....Care Riverside Club, Penns Grove, N. J.
 SCHUYLER, ARENT H., Snyder Electric Furnace Co., 53 W. Jackson Boulevard,
 Chicago, Ill.
 SELBIE, CHARLES C.....Care Colorado Min. Co., Aroroy, Masbate Island, P. I.
 SEMPLE, R. A.....702 Buffalo Ave., Niagara Falls, N. Y.
 SEVERY, C. L.....Care Carter Oil Co., 307 Central Bank Bldg., Tulsa, Okla.
 SHAFFER, WILLIAM B.....Nazareth, Pa.
 SHEARMAN, WILLIAM H.....55 Wall Street, New York, N. Y.
 SIMON, TREVOR B.....316 Market St., Scottsdale, Pa.
 SIRDEVAN, W. H.....118 E. Water St., Olean, N. Y.
 SLAUGHTER, B. G.....Vice-Prest., Tennessee Copper Co., Copperhill, Tenn.
 SMITH, LLOYD B.....Box 305, Ardmore, Okla.
 SMITH, LYON.....621 West 6th St., Reno, Nev.
 SOMMERVILLE, A. O.....Penn Coal & Coke Corp., Patton, Pa.
 SPEAK, S. J., Hooper.....Speak & Co., 5 London Wall Bldgs., London, E. C.
 SPENCE, HAROLD, C. E.....Duncan, Ariz.
 STANLEY, J.....Great Brampton, Hertford, England.
 STOIBER-ROOD, L. A., Care Fifth Avenue Bank of N. Y.,
 530 Fifth Ave., New York, N. Y.
 SUVERKROP, E. A.....305 W. Second St., Dayton, O.
 TAYLOR, JAMES, Care Mr. F. A. Smith, 82 Marlcliffe Road, Sheffield, England.
 TAYLOR, W. W.....124 Howard Ave., New Haven, Conn.
 THOMPSON, A. P.....Care Utah Apex-Mining Co., Bingham Canyon, Utah.
 THOMSON, S. C.....120 Broadway, New York, N. Y.
 TICKNER, FRANK W.....Gen. Mgr., Valley Mould & Iron Co., Sharpsville, Pa.
 TRENGROVE, S. R.....2936 Halldale St., Los Angeles, Cal.
 TURNER, A. A., Mgr.....Daggett Red. Co., & Goldstone Min. Co., Barstow, Cal.
 TURNER, R. CHESTER.....Brunswick Mine, Grass Valley, Cal.
 TUTTLE, ARTHUR L.....Care Tennessee Copper Co., Copperhill, Tenn.
 VAN CAMPEN, FRANK RUMSEY.....Kennecott Copper Corp., Latouche, Alaska.
 VAN HORN, E. A.....Supt., Lytle Coal Co., Minersville, Pa.
 VAN RENSSELAER, A. M.....21 Linden Place, New Rochelle, N. Y.
 VAN VALKENBURGH, R. D.....Delaware & Hudson Coal Co., Scranton, Pa.
 VERNON, J. A.....44 W. 9th St., New York, N. Y.
 VIGEON, E. C., Met. Mgr., Bade Metal and Chemical Co., Ltd.,
 Hebburn-on-Tyne, England.
 WAGONER, LUTHER.....24 Calle Cuba, Havana, Cuba.
 WALKER, R. T.....Techatticup Mine, Nelson, Clark Co., Nev.
 WALSH, T. D.....Care Nevada Arizona Mines Co., Hackberry, Ariz.
 WANG, C. Y.....Panoff Garden, Rue de Saigon, Hankow, China.
 WENDLER, HERMAN J., Vice-Prest., International Investment Syndicate,
 1119 Investment Bldg., Los Angeles, Cal.
 WEST, H. E.....Box 206, R. F. D. 1, Santa Barbara, Cal.
 WHERRY, H. P.....University Club, Chicago, Ill.
 WHITE, LEONARD L., Care New York & Honduras Rosario Min. Co.,
 San Juancito Honduras, C. A.
 WILKENS, H. A. J.....120 Broadway, New York, N. Y.
 WISER, OBA, Met. Engr.....Chino Copper Co., Hurley, N. M.
 WOODS, CLARENCE.....Ocean Star Mine, Washington, Cal.
 WOOLMER, H. C., Genl. Mgr., The Spasky Copper Mines, Ltd., 17 Mesnitky,
 Moscow, Russia.
 WRAMPMEIER, E. L. S.....701 First National Bank Bldg., San Francisco, Cal.
 WROTH, J. S.....Room 3533, 120 Broadway, New York, N. Y.
 YEN, C.....137 Hubbell Ave., Houghton, Mich.
 ZELLER, H. P., Supt., By-Product Coke Plant, Toledo Furnace Co., Toledo, Ohio.

CHANGES OF ADDRESS OF MEMBERS NOT YET CONFIRMED

- BROWN, FRED C.....Atlanta, Ida.
 BURCHELL, H. C.....Windsor, N. S., Canada.

CLARK, FREDERICK P.	4311 Barring Ave., East Chicago, Ill.
CONNER, E. T.	Union National Bank Bldg., Scranton, Pa.
FRANKE, EVIL A.	Minocqua, Wis.
GOODRICH, H. B.	Randol Hotel, Ardmore, Okla.
HAMMOND, J. H.	150 Broadway, New York, N. Y.
IMHOFF, ALEXANDER,	Care Chamber of Mines & Oil, 224 South Spring St., Los Angeles, Cal.
KEEP, GLENN A.	1277 E. 1st South St., Salt Lake City, Utah.
PORTER, J. C., Supt.,	Mina San Fernando, Hoyo de Manicaragua, Santa Clara Province, Cuba.
REDFEARN, ALBERT M.	1069 John R. St., Detroit, Mich.
RHOADS, ALBERT E.	Care Lackawanna Club, Lackawanna, N. Y.
RICE, E. ROCHE.	Mission Garage, El Paso, Tex.
ROBESON, JACOB H.	514 Boston Bldg., Denver, Colo.
ROSSI, AUGUSTE J.	Box 745, Niagara Falls, N. Y.
SEARS, MORTIMER A.	323 New P. O. Bldg., Denver, Colo.
WEINBERG, E. A.	15th St. and Union Sq. West, New York, N. Y.

MEMBERS' ADDRESSES WANTED

Name.	Last address of Record from which Mail has been Returned.
ALLEN, GLENN L.	Univ. of Utah, Box 88, Salt Lake City, Utah.
COPELAND, ROBERT NATHANIEL,	Care Compania Estanifera de Llallagua, Llallagua, Bolivia, So. Amer.
DURANT, H. T., Care Min. and Met. Club,	3 London Wall Bldg., London, E. C., England.
GRANT, ULYSSES S., IV.	Westmorly Court, Cambridge, Mass.
GREENAN, JAMES O.	Sarita Min. Co., Masonic, Cal. via Sweetwater, Nev.
HANLON, JOHN EDWARD.	Timmins, Ont., Canada.
LOERPABEL, W. HARRISON.	116 Ellison St., Lead, So. Dak.
MARRIOTT, ALEXANDER D., JR.	P. P. I. E., Box 6, San Francisco, Cal.
PARRISH, S. F.	1400 Union Trust Bldg., Los Angeles, Cal.
SANDIFER, HARMER C.	Box 15, Bis., Mexico City, Mex.
SOPER, RALPH H.	Care E. A. Soper, 1401 Cowper St., Palo Alto, Cal.
THOMAS, EDMUND.	Weaver Mine, Gibson, N. Mex.
WASH, EDWIN R.	Mascot Copper Co., Dos Cabezas, Ariz.
WRIGHT, CLARK W.	Little River Drainage District, Delta, Mo.

NECROLOGY

The deaths of the following members was reported to the Secretary's office during the period Jan. 10, 1916 to Feb. 10, 1916.

Date of Election.	Name.	Date of Decease.
1914	*Clark, Patrick F.	June,—, 1915.
1914	*Hase, Herman Carl.	Jan. 11, 1916.
1876	*Jennings, Edward P.	Dec. 31, 1915.
1888	*Kada, Tsuichi.	Nov. 29, 1915.
1906	*Morse, Bryan K.	Jan. 12, 1916.
1907	*Pringle, Charles A.	Jan. 11, 1916.
1914	*Rose, Charles (Clemons).	July 17, 1915.
1914	*Wallace, W. J.	Jan. 11, 1916.
1915	**Watson, Charles R.	Jan. 11, 1916.

*Member.

**Life member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.
 DAVID H. BROWNE, *Chairman*. JOHN H. JANEWAY, *Vice-Chairman*.
 F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
 LEWIS W. FRANCIS. P. A. MOSMAN, *Treasurer*. BENJAMIN B. LAWRENCE.

Boston

Meets first Monday of each winter month.
 HENRY L. SMYTH, *Chairman*. ALFRED C. LANE, *Vice-Chairman*.
 HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
 A. H. EUSTIS, F. G. STANTIAL.

Columbia

Holds four sessions during year. Annual meeting in September or October.
 STANLEY A. EASTON, *Chairman*. FREDERIC KEEFER, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.

Puget Sound

Meets second Saturday of each month.
 GLENVILLE A. COLLINS, *Chairman*. H. L. MANLEY, *Vice-Chairman*.
 AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
 I. F. LAUCKS, JOHN N. POTT.

Southern California

SEELEY W. MUDD, *Chairman*. C. COLCOCK JONES, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
 RALPH ARNOLD. A. B. W. HODGES.
 A. B. CARPENTER. WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman*. F. H. BOSTWICK, *Vice-Chairman*.
 P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
 G. A. KENNEDY. M. S. MACCARTHY.

Montana

FRANK M. SMITH, *Chairman*. JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST. W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.
 T. A. RICKARD, *Chairman*. W. H. SHOCKLEY, *Vice-Chairman*.
 C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
 E. A. HERSAM. H. W. YOUNG.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.
 CHARLES F. HUBER, *Vice-Chairman*. EDWIN LUDLOW, *Vice-Chairman*.
 W. J. RICHARDS, *Vice-Chairman*. ARTHUR H. STORRS, *Vice-Chairman*.
 PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

ARTHUR THACHER, *Chairman*. R. A. BULL, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 C. J. ADAMS. H. A. BUEHLER. HERBERT A. WHEELER.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Flotation Concentration at Anaconda, Mont.

BY FREDERICK LAIST,* B. S., AND ALBERT E. WIGGIN,† B. S., ANACONDA, MONT.

(Arizona Meeting, September, 1916)

I. EXPERIMENTAL FLOTATION CONCENTRATION

INTRODUCTION

EARLY in 1914 it was decided to test, on a fairly large scale, the treatment by flotation of Anaconda slime and mill tailing. For this purpose a standard-type Minerals Separation machine was installed at the Washoe Reduction Works during May and June, 1914. This was followed by the installation of a full-size Callow pneumatic machine plant. Experiments were also made, on a smaller scale, with the Froment, the Towne, the Fields, and the Anaconda flotation machines. The last-named machine was developed at this plant. In addition to the tests made in the standard-type Minerals Separation machine some tests were made using a Minerals Separation machine of the sub-aeration type.

During the series of experiments a large variety of oils was tested. Experiments were also conducted using both round-table feed and tailing to determine whether it would be better to displace the round tables by flotation for the treatment of the slime, or to supplement the round tables by flotation of the round-table tailing.

A series of tests was also made on the treatment of the mill tailing by grinding followed by flotation to determine the relative merits of flotation and leaching for the treatment of this product. In addition, flotation tests were made on mixtures of mill tailing and slime.

The round-table feed referred to above is the total slime from the mill. It contains about 35 per cent. colloidal solids and approximately 90 to 95 per cent. of the total solids will pass through 200 mesh (0.067 mm.). It assays from 2.3 to 2.6 per cent. Cu.

The mill tailing referred to above is the total discard from the mill, exclusive of the slime. It is all finer than 2 mm. and about 90 to 95 per cent. will remain on 0.25 mm. It assays about 0.60 per cent. Cu.

A brief summary of the experimental flotation results follows:

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Preliminary Tests

A series of tests was first carried out to determine roughly the best conditions for flotation, using the standard Minerals Separation machine and treating round-table feed: The following reagents were tested either alone, or in combinations: Turpentine, crude oil, cresylic acid, stove oil, tar oil, Carolina oil of tar, argols, sludge acid, fuel oil, wood creosote, and sulphuric acid. In some of these tests sulphuric acid was used and in others it was omitted. Also, the effect of the temperature of the pulp upon the flotation results was tested by heating to various temperatures.

As these tests were merely preliminary, no record was kept of the amount of reagents used. It was conclusively proved, however, that the best combination of reagents was sludge acid, wood creosote, stove oil, and sulphuric acid. Fortunately, of all the reagents tested, these happened to be the cheapest. It was also proved that the addition of sulphuric acid to the pulp was of decided advantage in the treatment of the slime. In two successive tests in which sludge acid, wood creosote, and stove oil were used, the tailing assayed 1.25 per cent. Cu when no acid was used and 0.3 per cent. Cu when acid was used. Since these tests were made we have omitted the use of stove oil.

A. TESTS WITH STANDARD MINERALS SEPARATION MACHINE

This machine, with the accessory apparatus, was installed in a separate building, south of the round-table plant. It had 16 agitator compartments, each 2 ft. square, and 14 spitzkästen, and was of the standard Minerals Separation design. This machine is known by us as M. S. Machine No. 1. The agitators were of the standard Minerals Separation type, the impellers being 18 in. in diameter and the agitators making 265 r.p.m. This gave the impellers a peripheral speed of 1,245 ft. per minute. The machine required 45 to 55 hp., including motor and belt transmission loss, when operating under a full load of slime pulp.

We wish at this point to express our appreciation of the able manner in which the experimental work on the Minerals Separation machine was carried out by George A. Chapman, and staff, of the Minerals Separation Co.

The first products to be tested were the round-table feed and tailing.

1. Treatment of Round-Table Feed and Tailing

(a) *Round-Table Feed. Normal Tonnage (60 Tons). Pulp Heated to 70° to 90°F.*—The reagents used in Periods 2 and 11 were sludge acid kerosene (M. S. 37), crude wood creosote (M. S. 33), stove oil (M. S. 8), and sulphuric acid. The reagents in Period 20 were the same, except

that the wood creosote was obtained from the Pensacola Tar & Turpentine Co., instead of from the Cleveland-Cliffs Co. The creosote from the Cleveland-Cliffs Co., known as M. S. 33, is a derivative from hard pine, while that from the Pensacola Co. is derived from a soft pine. The sludge acid kerosene is a byproduct from the distillation and purification of kerosene, and that known as M. S. 37 is derived from California crude oil by the Union Oil Co. The stove oil, known as M. S. 8, is from the Standard Oil Co. of California, and seems to be largely kerosene.

Period 2, in which the pulp was heated to an average temperature of 89°F., gave a tailing running 0.23 per cent. Cu, with a fair grade of concentrate.

Period 11, in which more creosote was used, all other conditions remaining as in Period 2, gave a tailing assaying 0.27 per cent. Cu, with a slightly better grade of concentrate. It should be noted, however, that it is not characteristic of creosote to give a clean concentrate, it being rather a good "tailing" oil and the slight increase in the grade of concentrate should not be attributed to the increase in the amount of creosote used, but rather to a difference in operation of the machine.

Period 20, in which Pensacola creosote was used, shows an average tailing of 0.33 per cent. Cu, and a very clean concentrate. It should be noted that considerably less creosote was used than in Period 11. The decided increase in the grade of concentrate is not due to the substitution of Pensacola creosote for Cleveland-Cliffs creosote, but rather to differences in operation.

(b) *Round-Table Feed. Pulp not Heated.*—During Periods 3 and 10, the pulp was not heated, but treated at a temperature of 50°F. The tests were made during the summer months; it should be noted that during the winter months the temperature of the pulp would be about 40°F., with a minimum of about 35°F. During Period 3, the tailing assayed 0.32 per cent. Cu, with a low-grade concentrate, and during Period 10 the tailing averaged 0.42 per cent. Cu, with a higher grade concentrate.

(c) *Round-Table Feed. Pulp Heated to 125°F.*—Heating the pulp to a temperature of 125°F. (Period 4) gave a tailing of 0.21 per cent. Cu with a fair grade of concentrate. As Period 2, in which the pulp was heated to 90°F. gave a 0.23 per cent. Cu tailing it is evident that heating beyond 90°F. does not pay. In fact, a careful study of the tabulated results would indicate that it does not pay to heat above about 75°F. The cost of heating the slime pulp for flotation is about 1c. per ton of dry slime for every 2½°F.

(d) *Round-Table Tailing. Pulp not Heated.*—The treatment of the round-table tailing, after dewatering to a density of 11.5 per cent. solids (Period 5), gave a tailing of 0.25 per cent. Cu with a low-grade concentrate. It was thought that possibly the net recovery would be sufficiently more to justify the retention of the round tables if the tailing

from the round table instead of the feed to the round table were treated by flotation.

(e) *Round-Table Tailing. Pulp Heated to 106°F.*—It is evident from the result of this test (Period 6) that the heating in the case of the round-table tailing was of no benefit.

(f) *Round-Table Feed. High Density of Pulp. Pulp Treated to 96°F.*—The density of the round-table feed was increased to 22.5 per cent. solids by a second dewatering (Period 7). As had been expected, the increase in density considerably decreased the oil consumption without decreasing the recovery of copper, and of course would also decrease the expense of heating the pulp.

(g) *Round-Table Feed. High Density of Pulp. High Tonnage (105 Tons). Pulp Heated to 96°F.*—It was thought that by increasing the density of the pulp it might be possible to treat the same volume of pulp per machine unit as with the lower density, and in that way increase the tonnage of solids treated. The tests under Period 8, however, show that any considerable increase in the amount of feed to the M. S. No. 1 machine over 60 tons results in a decided increase in the tailing loss.

(h) *Round-Table Feed. High Density of Pulp. High Tonnage. Pulp not Heated.*—The value of heating the pulp in the case of high-density feed pulp is brought out by this test (Period 9), where the tailing jumped to 1.28 per cent. Cu with the absence of heat.

(i) *Round-Table Tailing. High Pulp Density. Pulp Heated to 70°F.*—This test (Period 12) gave a tailing of 0.45 per cent. Cu, and a low-grade concentrate.

(j) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to 80°F.*—This test (Period 13A) gave a good tailing assaying but 0.20 per cent. Cu. This would seem to indicate that too high a pulp density in the treatment of the round-table tailing is a decided disadvantage.

(k) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to 80°F.*—In this test (Period 13B) no acid was used, and as a result the tailing went up to 0.96 per cent. Cu. Note also the low silver content of the concentrate.

(l) *Round-Table Tailing. Low Pulp Density. Pulp Heated to 70° to 80°F.*—In (Periods 14A, B, C, and D) the round-table tailing pulp was not dewatered at all, but treated directly at a density of about 6 per cent. solids. The tailing was good, ranging from 0.16 per cent. Cu to 0.22 per cent. Cu, for the four tests. During Period 4B, crude turpentine, M. S. 14, from the Georgia Fine Products Co., was used in place of Cleveland-Cliffs creosote; during Period 14C, crude wood creosote, No. 400, from the Pensacola Tar & Turpentine Co., was used, and during Period 14D, crude wood creosote from the Crichton Pine Products Co., Ala., was used. All of these oils gave satisfactory results. The increased amounts of these oils used during these tests were un-

doubtedly due to the low density of the pulp, and to the low tonnage treated.

(m) *Round-Table Feed. Low Pulp Density. Low Tonnage (40 to 50 Tons). Pulp Heated to from 70° to 80°F.*—It was thought that the low tailing produced during Periods 14A, B, C, and D was due to the low tonnage treated, and possibly to the low pulp density, as well as to the fact that round-table tailing was being treated. Accordingly, the tests under Periods 15A, B, C, D and E were made and in these a low tonnage of round-table feed was treated with a low pulp density. These tests gave fully as good results from the recovery standpoint in a considerably better grade of concentrate as those made on the round-table tailing under similar conditions.

During Period 15A, wood creosote from the Crichton Pine Products Co. was used. During Period 15D, no creosote was used and the tailing assay increased from about 0.20 per cent. Cu to 0.32 per cent. Cu. During Period 15E, some creosote from the Rocker Timber Treating Plant, near Butte, was used, but gave a high tailing—0.42 per cent. Cu. During Period 15B, tar creosote from the Butte Gas Co. was used, with apparently good results, although the test was too short to be taken as conclusive proof of the value of this creosote as a flotation oil.

(n) *Round-Table Feed. Low Tonnage (50 Tons). Pulp Heated to 70°F.*—The tests (Period 16A) gave a tailing running 0.21 per cent. Cu, and a fair grade of concentrate.

(o) *Round-Table Feed. Pulp Heated to 70°F. High Speed of Agitators.*—It was thought that by increasing the speed of the agitators the amount of feed treated might be increased without decreasing the recovery. The first nine agitators were used and their speed was increased from 265 r.p.m. to 363 r.p.m. The test (Period 16B) was of short duration, as the machine would have racked itself to pieces if operated very long at this high speed. While the test is too short to be conclusive, the indications are that the increased speed was a detriment rather than an advantage. It is thought that there is a critical speed at which the agitators should be run for the most efficient operation. If we go above this speed the power increases much faster than the capacity, and if we go below it we do not get a proper mixing of oil and acid with the pulp. With the agitators running at this speed, the nine cells required 56 hp. including motor and belt transmission loss.

(p) *Round-Table Tailing. Low Tonnage (40 Tons). Pulp Heated to 70°F. Low Pulp Density.*—These tests (Periods 17 and 18) show average tailings of 0.22 per cent. Cu and 0.26 per cent. Cu respectively. The concentrate is rather low grade.

(q) *Round-Table Feed. High Tonnage (90 Tons). Pulp Heated to 70°F.*—The tests under Period 19 gave a high tailing, 0.43 per cent. Cu, and a good grade of concentrate. Some wood creosote from the

Pensacola Tar & Turpentine Co. was used together with that from the Cleveland-Cliffs company. The high tailing is due, however, entirely to overloading the machine.

Under Period 23 the tonnage treated was just as high, the grade of concentrate only a little lower, and yet the tailing is good, 0.25 per cent. Cu. This apparent discrepancy between these two tests is explained, it is thought, by the fact that the slime produced in the mill was considerably more granular during the tests under Period 23 than during those under Period 19.

(r) *Round-Table Feed. Low Tonnage (40 Tons). Pulp Heated to 70°F.*—In this test (Period 21), the tailing averaged 0.25 per cent. Cu and the concentrate was medium grade. No Cleveland-Cliffs creosote was used, all the creosote used being from the Pensacola company. The tailing is practically no better with this low tonnage (40 tons) than under Period 23 with the high tonnage (90 tons).

(s) *Round-Table Feed. High Tonnage (90 Tons). Air Used in Last Spitzkasten.*—It was thought that the introduction of air in the bottom of the last spitzkasten might increase the capacity of the machine and at the same time maintain the same mineral recovery. At first it seemed as though the introduction of the air (Period 22) had resulted in a considerable increase in the capacity of the machine, but when the tests under Period 23 were made, in which the air was omitted, it was found that it was not the introduction of the air which had increased the tonnage.

(t) *Conclusions.*—1. The economic capacity of the M. S. No. 1 machine when treating slime as produced from the mill at present (May 1, 1915) seems to be from 80 to 90 tons.

2. The best combination of reagents for the treatment of slime seems to be sulphuric acid, kerosene sludge acid, wood creosote and stove oil. There is some question as to the real value of the stove oil. Its principal function seems to be to make a more compact froth.

3. It would not be economical to retain the round tables as the recovery by treating the slime directly by flotation is just as high as by retaining the round tables and treating the round-table tailing by flotation. The grade of concentrate would probably be the same in either case, but any difference would be in favor of treating the round-table feed directly by flotation. The heating of the round-table tailing pulp, on account of its low density, would increase the cost of the flotation.

4. In treating the round-table feed directly by flotation, the resulting tailing should assay 0.30 per cent. Cu, or less, with a concentrate carrying not over 40 per cent. insoluble. Possibly the concentrate can be made much cleaner with no sacrifice in the recovery.

5. It is thought that the best circuit density for the slime pulp for flotation treatment is about 12 per cent. solids.

6. It is thought that about 70°F. will be found to be the most economical temperature at which to keep the pulp.

7. Acid seems to be absolutely essential to the successful treatment by flotation of our slime.

8. The addition of air in the last spitzkasten is of no advantage.

9. Any considerable increase in speed of the agitators above a peripheral speed of about 1,300 ft. per minute seems to be disadvantageous.

2. *Treatment of Mill Tailing after Grinding through 60 Mesh*

These tests were made in the M. S. No. 1 machine. Mill tailing from Sections 7 and 8 of the concentrator were elevated and then dewatered. The dewatered tailing was then crushed through 60 mesh (0.25 mm.), in either a Hardinge mill 10 by 4 ft., or a tube mill 8 by 12 ft. The grinding mills were operated in closed circuit with a Dorr classifier, the overflow of the classifier being the final product of the system and going to the flotation plant for treatment.

Following is a screen sizing test of the average final product produced during the test on the Hardinge mill, and is typical of the flotation feed:

Screen Sizing Test on Dorr Classifier Overflow, or Feed to Flotation Machine

Screen Size		Cumulative Per Cent., Solids
Square Mesh	Aperture, Mm. Square	
+16	1.180	0.3
+24	0.730	1.3
+40	0.430	3.0
+60	0.260	5.8
+80	0.210	12.8
+110	0.130	38.5
+130	0.110	42.3
+160	0.085	54.8
+200	0.076	59.3
+240	0.063	62.8
-240	0.063	37.2

(a) *Preliminary Tests.*—These tests (Period 24) were started immediately after putting the Hardinge mill in operation. At first no sulphuric acid was added, and the pulp was not heated. It would be well to note here that the sludge acid kerosene contains from 50 to 60 per cent. sulphuric acid, so that when this oil is used we could not have a non-

acid pulp. We found, however, that the use of acid in addition to that contained in the sludge was of advantage. Some very low tailings were produced during these preliminary tests, but the concentrate was very low grade. It seemed to be of decided advantage to add the oil ahead of the grinding mill, the latter apparently making an ideal agitator.

However, as soon as we began to heat the pulp ahead of the mill, we found that the acid in the oil and the small amount of copper sulphate formed began to corrode the iron work of the mill. This would not be a serious matter if we used a pebble, or silex-block lining, with pebbles for grinding, but we early abandoned the idea of using silex, or pebble linings, and there was a good chance that as a grinding medium we would find iron balls superior to pebbles. This, of course, precluded the use of any acid, or acid oil, ahead of the grinding mill, especially if the pulp was heated.

It is barely possible that we might find the action of the sludge acid kerosene in a cold circuit so slight that it could be neglected, even though we used steel lining and steel balls. It would be a decided advantage to add the oil ahead of the grinder.

(b) *Low Tonnage (125 Tons). Pulp Heated to 70°F. Oil and Acid added in M. S. Machine. Acid Circuit.*—The tailing produced during this period (No. 25) averaged 0.07 per cent. Cu. The concentrate was too siliceous, averaging 40.1 per cent. insoluble. The only reagents used were sulphuric acid and sludge acid kerosene. It should be noted that the product treated during this period contained only 3.1 per cent. on 80 mesh (0.20 mm.).

(c) *High Tonnage (187 Tons). Same Conditions as under (b), except that Flotation Machine was Divided into Two Parts, Making a Primary and a Secondary Machine, the Secondary Machine being used to Clean the Concentrate from the Primary Machine.*—The flow sheet was as follows: The feed pulp was put into the sixth agitator cell, thus making the last 11 cells act as a primary machine. The concentrate from the first six boxes of the primary machine was returned to the first three boxes of the original machine through the two pre-agitators. The overflow from the remaining five boxes of the primary machine was returned to the original feed. The last cell made the usual tailing. The secondary machine made a final concentrate and a middling which was returned to the circuit. The chief reagents used were sludge acid kerosene and sulphuric acid. Small amounts of creosote were used during part of the test, but it seemed to be of no particular advantage. During two days, sludge acid kerosene from the Standard Oil Co. of California was used with good results.

The tailing during this period (No. 26) averaged 0.11 per cent. Cu, and the concentrate averaged 26.9 per cent. insoluble. It is thought that this is about typical of the results we may expect from a commercial

installation. It should be noted that 14.4 per cent. of the flotation feed remained on an 80-mesh screen (0.2 mm.). Finer grinding would undoubtedly result in a lower tailing assay, but we must, of course, set against the increased copper recovery the cost of the finer grinding.

It would be well to call attention to the fact that all averages shown in these flotation tabulations are arithmetical and not geometrical. It was found that the arithmetical average practically checked the geometrical in the test cases, and as the geometrical averages would have involved a great deal of extra work it was considered not worth while to average the results geometrically. For example, Period 26 was averaged geometrically and gave a tailing assay of 0.109 per cent. Cu against the arithmetical average of 0.11 per cent. Cu. The concentrate averaged 7.777 per cent. Cu geometrically, and 7.82 per cent. Cu arithmetically. As the daily variations in tonnage and assays are considerable during this period, we would expect to find the arithmetical and geometrical averages diverging more than during the other periods in which the variations are not so great.

(d) *Low Tonnage. (124 Tons). Same 'Conditions as under (c), except Turpentine (M. S. 14) added ahead of Grinding Mill, and no Sludge Acid Kerosene used. Pulp Heated to about 70°F.*—In this test (Period 27) it was endeavored to take advantage of the grinding mill as an agitator and for this reason a neutral reagent, turpentine, was used. The tailing is high, 0.12 per cent. Cu, considering the comparatively low tonnage of feed and the fact that only 5.2 per cent. of the feed was coarser than 80 mesh. The same flow sheet was used as during Period 26.

(e) *Low Tonnage (106 Tons). Oil added ahead of Grinding Mill. Non-Acid Circuit. Pulp Heated to about 75°F. Various mixtures of crude turpentine, creosote and pine oil used during these tests. The mixtures contained from 60 to 70 per cent. crude turpentine; 21 to 32.5 per cent. creosote; 2.5 to 15 per cent. pine oil.*—The tailing produced during this period was fairly good, averaging 0.09 per cent. Cu, but the grade of concentrate was not good, averaging 41.5 per cent. insoluble.

(f) *Conclusions.*—1. Although not definitely demonstrated, it is thought that the economical capacity of the M. S. No. 1 machine when treating sand tailing crushed through 60 mesh is about 175 to 200 tons per 24 hr.

2. The best combination of reagents seems to be sludge acid kerosene and sulphuric acid. However, a mixture of creosote, turpentine, and pine oil, in a non-acid circuit gave good results also. The non-acid circuit, however, seems to require more delicate adjustment and more careful attendance than the acid circuit.

3. The grinding mill makes an ideal agitator, and it is of decided advantage to add the oil ahead of the grinders.

4. The treatment of the mill sand tailing ground through 60 mesh

should result in a tailing assaying not over 0.10 per cent. Cu and a concentrate carrying not over 30 per cent. insoluble.

5. It is thought that the best density of pulp is from 25 to 30 per cent. solids.

6. Heating of the pulp to about 70°F. seems to be of advantage, although there is a possibility that this heating may be dispensed with during the summer months without any injurious results.

7. Acid seems to be beneficial but it is not of as much importance as in the treatment of the slime.

3. Treatment of Mixture of Round-Table Feed and Mill Tailing after Grinding through 60 Mesh

These tests were made in the M. S. No. 1 machine. It was thought that it might be of advantage to mix the slime and the reground mill tailing for flotation treatment. The acid sludge kerosene, turpentine and the sulphuric acid used were added in the flotation machine. In some instances, various mixtures of coal tar (70 to 80 per cent.), creosote (17.5 to 22.5 per cent.), and pine oil (2.5 to 7.5 per cent.) were used with the sludge acid. These were added ahead of the grinding mill.

The average proportion of sand tailing to slime in the mixture treated was 75.7 to 20.1, or 3.8 to 1. In practice the proportion of production of tailing to slime is about 3 to 1; thus our mixture was somewhat deficient in slime. The concentrate produced, 34.1 per cent. insoluble, is of a good grade, but the tailing is high, 0.20 per cent. Cu. Theoretically, the tailing should have assayed about 0.15 per cent. Cu, assuming a 0.10 per cent. Cu tailing from the sand tailing and 0.30 per cent. Cu tailing from the slime.

Although this test was not conclusive, it was decided, from observation, that it is better to treat the slime and the sand tailing separately. Of course, the slime which is made in the grinding of the sand tailing is included in the sand tailing for treatment. This slime produced in grinding the tailing is much lower grade and more siliceous than the original mill slime.

B. TESTS WITH MINERALS SEPARATION MACHINE OF SUB-AERATION TYPE

1. Treatment of Round-Table Feed and Tailing

This machine was sent here by the Minerals Separation Co., and was set up and tested at its request.

The principal difference between this and the No. 1 machine is that air is introduced at the bottom of the agitator compartment. The agitator was of the Howard type, practically a "balanced" pump im-

PELLER run backward, and each shaft had two impellers, one placed above the other and separated by a stationary grid. There was also another grid above the top impeller. There were four agitator compartments and four spitzkästen. The agitators made 185 r.p.m. At first it was planned to suck air from the atmosphere by the action of the impellers. This did not work, however, and it was found necessary to force air into the cells under a few pounds pressure. The feed was introduced through the bottom of the first agitator cell. The laundering was arranged so that as many spitzkästen as desired could be sent to finished concentrate, the remainder going to middling and being returned to the machine. This machine required 47 hp. under a full load of slime, including motor- and belt-transmission loss.

Two tests were made, one treating round-table feed, and one treating round-table tailing. The results of these tests are shown in Periods 30 and 31 in the tables. It may be possible to develop an efficient machine of this type, but the particular machine sent here for testing was certainly not satisfactory.

A later test was made in which the spitzkästen were dispensed with and the froth was taken directly off the top of the agitator cells. Under these conditions, the four agitators seemed to create too much disturbance for the proper removal of the froth, and only two agitators, the first and third were operated. These modifications did not improve the work of the machine. The results of this test are shown below. This modified machine was called No. 2 A.

C. TESTS WITH CALLOW PNEUMATIC MACHINE

Tests made by Mr. Callow at his laboratory in Salt Lake on samples of our mill tailing ground through 40, 60, and 80 mesh, and of our slime, had given such promising results that it was decided to try out the Callow machine on a commercial scale. Accordingly, there was shipped here during September, 1914, five standard Callow cells, 2 by 8 ft., a Pachuca agitating tank and accessory apparatus, consisting of blower and sand pumps. This equipment was installed in the old 80-ton experimental leaching plant and was ready for operation the latter part of October.

In addition to the Pachuca agitator recommended by Mr. Callow, we built a set of two mechanical agitators. These agitators consisted of a tank about 10 ft. long by $2\frac{1}{2}$ ft. wide and $2\frac{1}{2}$ ft. deep, in which revolved a horizontal shaft carrying a set of paddles. These agitators were belt driven from one motor and required a total of 25 to 30 hp., including motor, belt, and countershaft power loss. The agitators seemed to work well and had a combined capacity of about 60 tons of slime per 24 hr.

Power readings made on the blower which furnished the air for the five Callow cells gave the following results:

Air Pressure, Pounds	Input to Motor, Hp.	Shafting, Hp.	Net to Blower,* Hp.
4	25.8	8.8	17.0
5	35.4	8.8	26.6
6	47.4	8.8	38.6

* Includes motor loss.

1. Treatment of Round-Table Feed and Tailing

(a) *Round-Table Feed. No Acid.*—During the first four days air agitation was employed, using the Pachuca tank. With a feed of 60 to 80 tons, the results were very poor. The air agitator did not have sufficient capacity. During the last five days of this period (No. 32) mechanical agitation was used and gave much better results.

(b) *Round-Table Feed. Air Agitation.*—This test (Period 33) which was of 16 days' duration gave an average tailing of 0.35 per cent. Cu, and a concentrate carrying 31.6 per cent. insoluble. The capacity per cell seems to be about 15 to 20 tons of slime per day. Sludge acid and sulphuric acid, with no creosote or stove oil seemed to give as good or better results as when creosote was used.

(c) *Round-Table Feed. Mechanical Agitation.*—This test (Period 34) of 27 days' duration, gave an average tailing of 0.30 per cent. Cu, and a concentrate assaying 34.2 per cent. insoluble. The average tonnage per cell was 20 tons per day. The principal oil used was sludge acid kerosene, together with sulphuric acid.

(d) *Round-Table Tailing. Mechanical Agitation.*—This test (Period 35), of four days' duration, gave an average tailing of 0.32 per cent. Cu, and a final concentrate of about 60 per cent. insoluble. (The rougher concentrate for the first two days should be included in the average final concentrate.)

(e) *Conclusions.*—1. On our slime, air agitation is not as satisfactory as mechanical.

2. The capacity of one standard Callow cell is about 15 to 20 tons of slime per day.

3. The Callow machine produces a clean concentrate but does not give as clean a tailing as the Minerals Separation machine.

4. The Callow machine is more sensitive and requires closer attention than the Minerals Separation machine.

5. The cost of repairs would probably be less on the Callow machine than on the Minerals Separation machine. This cost, however, is comparatively small for either machine.

6. The power required per ton treated in the Callow system is just about the same as that required in the Minerals Separation machine.

In all of these tests the original feed was divided among the Callow rougher cells, operating in parallel. As a rule, there was one cleaner cell operating also. When this was operating the concentrate from the rougher cells went to it, the cleaner making a final concentrate and a middling which was returned to the system. The rougher cells made the final tailing.

2. Treatment of Mill Tailing after Grinding through 60 Mesh

(a) *Preliminary Tests.*—During the first few shifts the mechanical agitators at the Callow plant were used, but it was soon found that they were not required—that the grinding mill gave sufficient and thorough agitation.

Sludge acid kerosene was the only oil used during this period, and was added ahead of the grinding mill. Sulphuric acid was added ahead of the flotation cells. The tailing for this period averaged 0.10 per cent. Cu, and the concentrate carried an average of 42.2 per cent. insoluble. The pulp was heated just ahead of the flotation cells (Period 36).

(b) *Sludge Acid Kerosene Added Ahead of Grinding Mill.*—The flow sheet was modified slightly during Period 37. Instead of sending the entire rough concentrate from the rougher cells to the cleaner cells, only the first half was sent to the cleaner, that coming off the half nearer the tailing end of the machine being returned as middling to the original feed. This resulted in giving the cleaner cell a richer feed and, in turn, enabled the cleaner to make a higher grade concentrate. This period shows some very good results, a tailing assaying 0.11 per cent. Cu and a concentrate carrying 27.9 per cent. insoluble. Here, again, is brought out the splendid agitation and mixing obtained in the grinding mill.

(c) *Neutral Circuit. No Acid. Turpentine Added Ahead of Grinding Mill.*—In this test (Period 38) no acid was used. Turpentine was added ahead of the grinder. The tailing averaged 0.16 per cent. Cu, and the concentrate carried 35.1 per cent. insoluble. It should be noted that in this test tonnage treated per cell was only 58.2 tons as against 75.6 tons in the preceding tests.

(d) *Neutral Circuit. Same Conditions as under (c) except Lower Tonnage per Rougher Cell.*—During this period (No. 39) the average tonnage treated per rougher cell was only 39.9 tons. The tailing was high, 0.18 per cent. Cu, and the concentrate low grade, 47.5 per cent. insoluble.

It is evident from the work in this and the preceding period that turpentine will not give us good results in a neutral circuit.

(e) *Mixtures of Coal Tar, Creosote and Pine Oil Added Ahead of Grinder. Neutral Circuit.*—The tests during Periods 36 and 37 were made under the supervision of Mr. Bernsen, a representative of Mr. Callow. These

tests show a tailing of 0.13 per cent. Cu, and a low-grade concentrate, — 45.3 per cent. insoluble. The results are not nearly so good as those obtained in the acid circuit.

(f) *Same Conditions as under Period 40, except the Rougher Cells Operated in Series instead of in Parallel.*—During this test the original feed went to No. 2 rougher and the tailing from No. 2 rougher was fed to No. 1 rougher, the latter making a final tailing. The concentrate from the No. 2 rougher went to the cleaner and that from No. 1 was returned to the feed as middling. The final tailing averaged 0.11 per cent. Cu, but the concentrate was rather low grade, running 38.7 per cent. insoluble.

(g) *Conclusions.*—1. The capacity of the standard Callow cell when treating ground mill tailing is about 75 tons per day.

2. No other agitation is required if the reagents can be added ahead of the grinding mill.

3. The use of acid seems to be of considerable advantage.

4. On account of utilizing the grinding mill as an agitator the Callow machine requires less power than the Minerals Separation machine.

5. The Callow machine is more sensitive and requires more attention than the Minerals Separation machine.

The work of the Froment and Towne, and the Fields flotation machines, was found to be unsatisfactory; that on the Anaconda cell was of short duration, no definite results being obtained.

D. MISCELLANEOUS FLOTATION TESTS

1. *Test of Various Kerosene Sludge Acid Oils*

During the experimental flotation work, tests were made on barrel lots of various sludge acid kerosenes, with the following results:

(a) *From Union Oil Co. of San Francisco.*—This is the sludge known as M. S. 37, used during practically all of the experimental work. It gives highly satisfactory results. Although exposed to the weather in iron drums during the winter months, it gave no trouble by becoming too viscous.

(b) *From Standard Oil Co. of San Francisco.*—This sludge acid gave just as good results in the flotation work as that from the Union Oil Co. This oil did not become viscous when exposed to freezing temperatures.

(c) *From Western Oil Co.*—One barrel of this oil was tested during February in the M. S. No. 1 machine when treating round-table feed. This oil did not give as satisfactory results as the two preceding oils. About one-third of the oil was left in the barrel as a solid residue.

(d) *From Midwest Refining Co.*—One barrel of this oil was tested during February in the M. S. No. 1 machine when treating round-table feed. The results obtained with this oil were not as good as those obtained when using the first two oils. At least one-half of this oil remained in the

barrel as a solid residue which we could not liquify even by using the steam coils.

(e) *From Producers Refining Co.*—We could not test this oil as it was solidified in the barrels and did not become liquid after standing for days at a temperature of 60°F., or better.

2. Test of Various Creosote Oils

During the experimental flotation work the following creosote oils were tested with the results shown:

(a) *Creosote from Cleveland-Cliffs Iron Co. of Marquette, Mich.*—This is a creosote obtained as a byproduct in the manufacture of charcoal from hard pine. This is known as M. S. No. 33, the number given to it by the Minerals Separation Co. This is the creosote that was used practically throughout the entire work with highly satisfactory results.

(b) *Creosote from Pensacola Tar & Turpentine Co. of Gull Point, Fla.*—We used a considerable quantity of this oil in our experimental work. It gives fully as good results as the Cleveland-Cliffs creosote. It is made from Georgia pine wood. This oil is known as No. 400 by the Pensacola Co. (see Periods 14C, 19, 20, 21, 22 and 23).

(c) *Creosote from Crichton Pine Products Co. of Alabama.*—A barrel of this oil was tested and gave good results (see Periods 14D and 15A).

(d) *Tar Creosote from Butte Gas Works.*—A test was made using a barrel of tar creosote from the Butte Gas Works, which gave fairly good results. The test was of too short duration to be conclusive, but it is thought that the tar creosote would give satisfactory results (see Period 15B). In testing the Callow machines a mixture of oil containing tar creosote from the Barrett Manufacturing Co. was used with good results (see Periods 40 and 41). In the laboratory tests we have found that the tar creosotes give good results.

(e) *Creosote from the Timber Treating Plant at Rocker.*—This creosote comes from the J. F. Lewis Co. of Moline, Ill. It did not give good results, although the test was too short to be absolutely conclusive. The oil contained a good deal of dirt and sediment (see Period 15E).

3. Test of Blankets for Callow Cells

Four blankets for Callow cells were received from Lane & Co., New York City. These blankets were numbered 1,038, 1,039, 1,057, and 1,058. In each case the blankets have a better distribution of air after a few hours' operation, due probably to a readjustment in the weaving after the blanket has been thoroughly wetted.

The blankets were tested, with the following results:

(a) *No. 1058, Single Thickness.*—Callow cells treating sand tailing.

The air distribution was uneven and if enough air was turned into chamber to keep sand moving on blanket, then there was violent boiling at points. Blanket not satisfactory for coarse (60-mesh) feed but might work fairly well on slime.

(b) *No. 1,058, Double Thickness.*—In this test the No. 1,058 blanket was doubled. The feed treated was slime. The air distribution was very good and the results practically the same as the regular Callow blanket supplied by the General Engineering Co. of Salt Lake.

(c) *No. 1,039.*—This blanket was tested on slime feed and gave an even distribution for all air pressures up to $5\frac{1}{2}$ lb.

(d) *No. 1,057.*—This blanket gave a very uneven distribution of air for all pressures. The pores seemed to close up unevenly when the blanket was wet.

(e) *No. 1,038.*—This blanket was tested on slime feed. Gave very good distribution of air for all pressures up to 4 lb. This blanket is thinner than the others and also thinner than the regular Callow blanket and requires less pressure to deliver the same volume of air as the others.

(f) *Résumé.*—No. 1,057 would not be satisfactory for our purposes.

No. 1,058, single, would not be satisfactory, except, possibly, on slime feed.

No. 1,058, double, would be satisfactory for either slime or sand, but would probably cost too much.

No. 1,038, entirely satisfactory for slime, and would probably be all right on sand.

4. Soluble Copper in Flotation Tailing

Samples of tailing pulp were sent to the laboratory and the soluble copper was determined.

The tailing from the round-table feed contained 0.141 gram of soluble copper per liter of solution. The tailing from the round-table tailing contained 0.084 gram of soluble copper per liter of solution.

5. Disintegration of Flotation Froth

In order to break up the froth in the flotation concentrate a saucer-shaped disk 3 ft. in diameter was constructed. This was revolved at a speed of 300 r.p.m. inside a circular tank. The concentrate pulp was fed on to the disk at the center and was thrown out against the sides of the tank. The impact broke up the froth to a great extent. No test was made on the disintegrator.

It was noted early in the experimental work that an elevator made a good froth breaker, the impact of the pulp discharge at the head of the elevator serving to break up the bubbles, to a large extent. A test made at the experimental Callow flotation showed that the rough concentrate pulp was reduced 75 per cent. in bulk in passing through an elevator.

E. CONCLUSION

The conclusions drawn from the foregoing tests were that the Minerals Separation machine was best adapted for the flotation work at Anaconda. Furthermore, that the most efficient reagents would be sludge acid kerosene, wood creosote, and sulphuric acid.

IV. DESCRIPTION OF REMODELED CONCENTRATOR AS ADAPTED TO FLOTATION

The concentrator at Anaconda, as remodeled for flotation, consists of eight sections, each of 2,000 tons per day capacity, giving a grand total of 15,000 tons per day, allowing for shutdowns, repairs, etc. All sections are alike with the exception of Section 1. In this section, Hancock jigs¹ are used in place of Evans jigs and tube-mills are used in place of Hardinge mills. Fig. 1 shows the flowsheet of Sections 2 through 8.

MILLING DIVISION

The ore is fed from the bins to a 2-in. round-hole shaking screen, the oversize going to a 12 by 24-in. Blake crusher. The product from this crusher is delivered to a 2-in. round-hole trommel, the oversize of which is sent to two 8 by 20-in. Blake crushers. The product from these crushers, together with the undersize from the 2-in. screens, is elevated and passed through 1-in. round-hole trommels. The oversize from this is treated in coarse Harz jigs, making a middling and a concentrate; the undersize is passed through $\frac{3}{8}$ -in. trommels, the oversize being treated in fine Harz jigs making a concentrate and a middling. All sections are alike up to this point. In Section 1, the undersize from the $\frac{3}{8}$ -in. trommel is screened on $1\frac{1}{2}$ by 12-mm. trommels, the undersize going to the Anaconda classifiers and the oversize to the Hancock jigs. The treatment of the products from this point is the same in all sections, except that Section 1 uses tube-mills in place of Hardinge mills for grinding, as noted previously. The undersize from the $\frac{3}{8}$ -in. trommel is screened through 4-mm. trommels, the oversize from these going to the double compound Evans jigs and the undersize going to $1\frac{1}{2}$ by 12-mm. trommels. The undersize from these trommels goes to the Anaconda classifiers, the oversize to double compound Evans jigs. The two sets of Evans jigs make a concentrate which goes to the dewatering bins and a middling which is ground for further treatment.

The concentrate from the coarse Harz jigs is dewatered and conveyed to bins. The middling is screened on a dewatering screen, the undersize together with the hutch product from the coarse Harz jigs going

¹ For comparative data on Hancock and Evans jigs, see *Trans.*, xlvii, p. 212) 1913).

INSTALLATION OF FINER GRINDING AND FLotation EQUIPMENT IN CONCENTRATOR

FLOW SHEET FOR SECTIONS Nos. 2-5

Original Ore (2000)

Ord Bin

4' x 24' Rod Shaking Screen

Overline (1100)

18' x 24' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Overline (1100)

3' x 30' Ball Grinder

Underline (900)

2' x 24' Rod Grindmill (8' x 8')

Note: Figures in Parenthesis
Show Tons of Solids per 24 Hours
Figures in Brackets
Show Gallons of Water per 24 Hours
Figures in Circles Show
Number of Machines

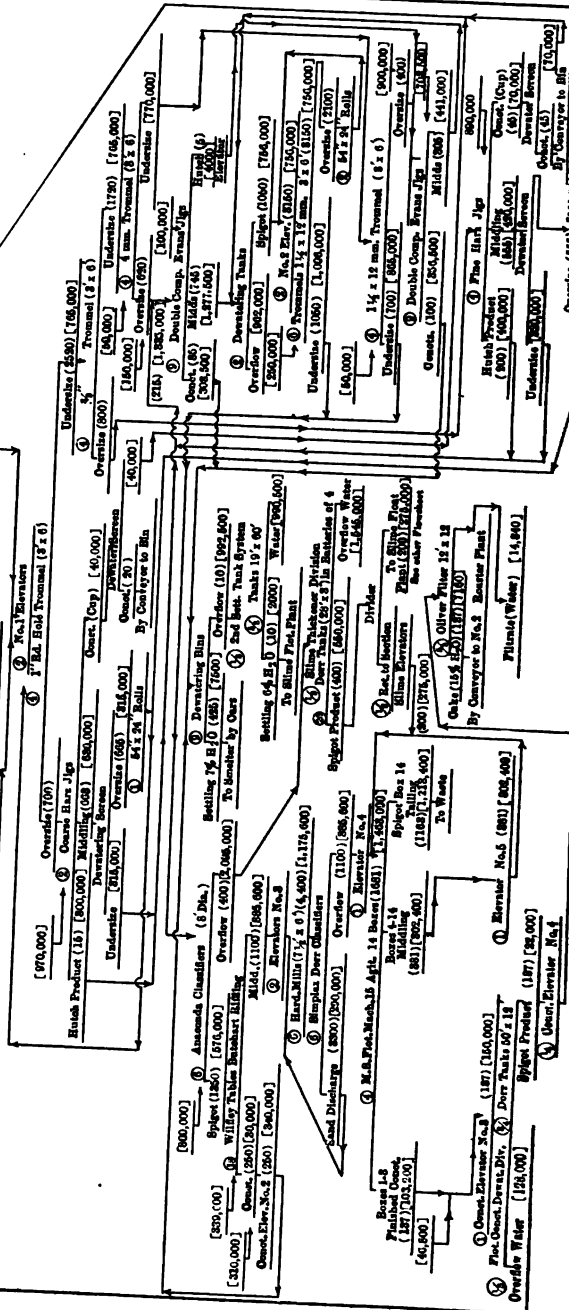


FIG. 1.

to the Evans jigs. The oversize is passed through rolls, 54 by 24 in., and thence back into the system ahead of the 1-in. round-hole trommels. The concentrate from the fine Harz jigs is sent to the bins. The middling is screened through a dewatering screen, the oversize going to 54 by 24-in. rolls and then back into the system ahead of the 1-in. round-hole trommels. The undersize of the dewatering screen together with the hutch discharge of the fine Harz jigs goes to the Evans jigs.

The concentrate from the Evans jigs is dewatered in bins to about 7 per cent. moisture, and sent to the smelter. The jig concentrate assays

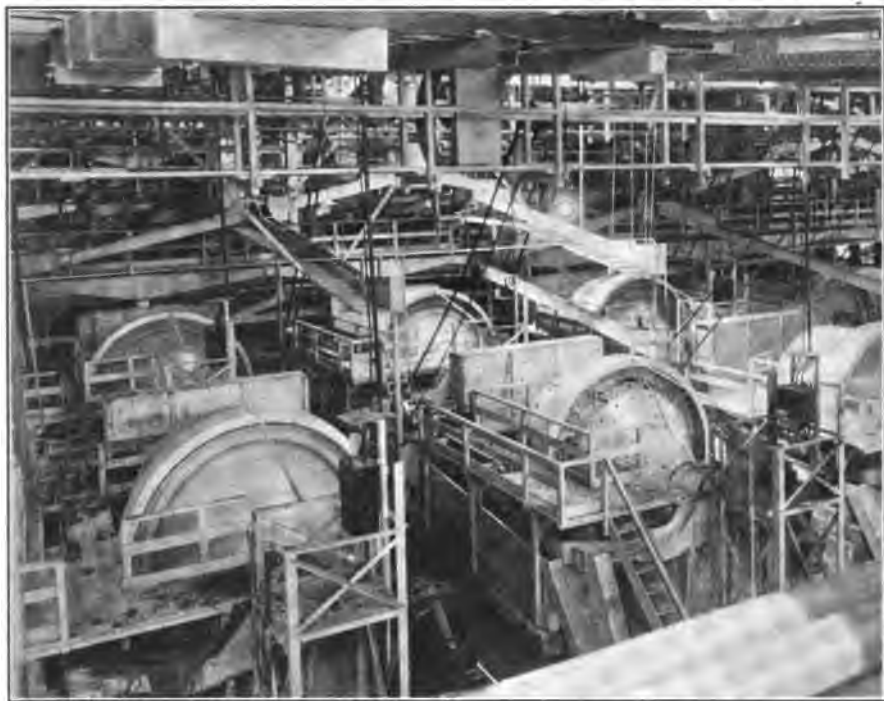


FIG. 2.—GRINDING DIVISION.

about 15 per cent. insoluble and 8 per cent. copper. The middling, together with the hutch product is dewatered in tanks and screened through $1\frac{1}{2}$ by 12-mm. trommels, the undersize from which goes to the Anaconda classifiers,² the oversize through 54 by 24-in. rolls, and back to the $1\frac{1}{2}$ by 12-mm. trommels.

The spigot from the Anaconda classifier is treated on 18 Wilfley tables, fitted with Butchart riffing, making a concentrate and a middling. These tables make a concentrate assaying 25 per cent. insoluble and a middling assaying 0.9 per cent. Cu. The concentrate is sent to the

² For description of Anaconda Classifier, see *Trans.*, xlv, p. 277 (1913).

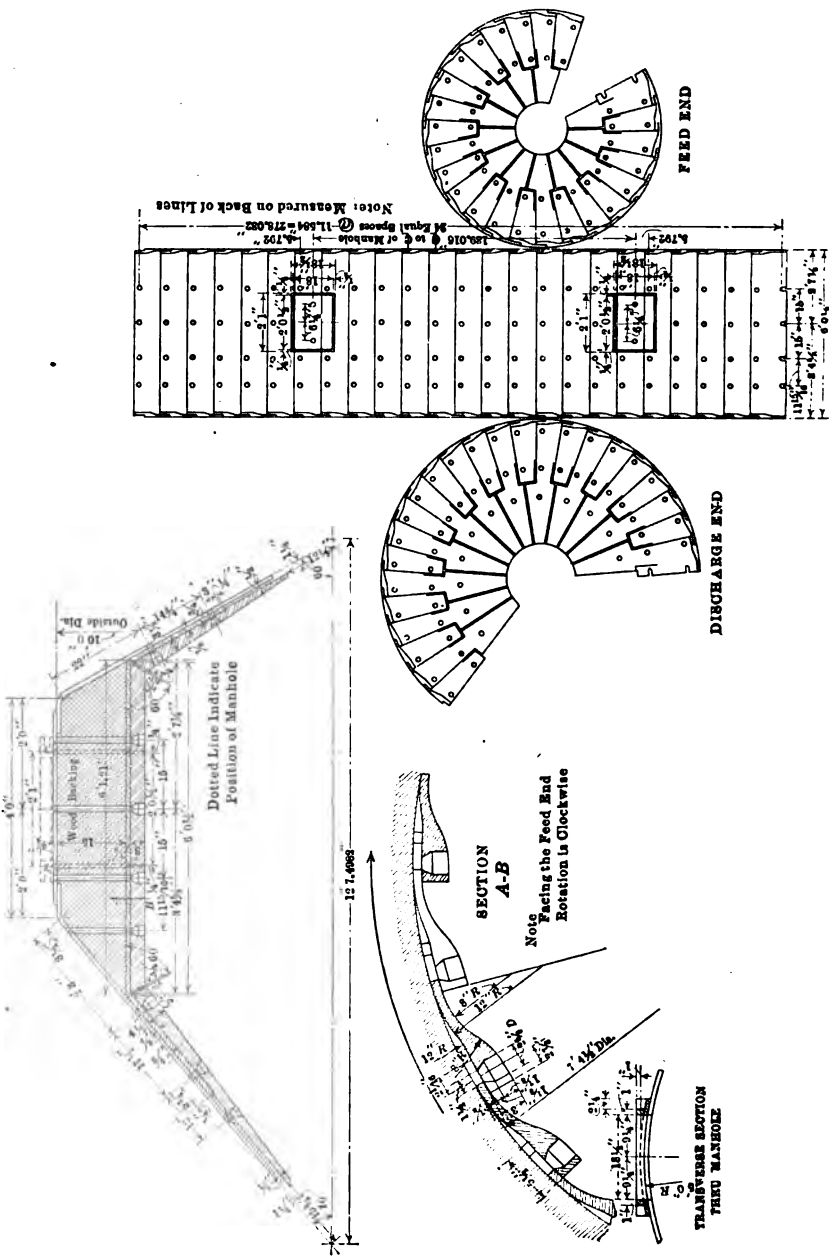
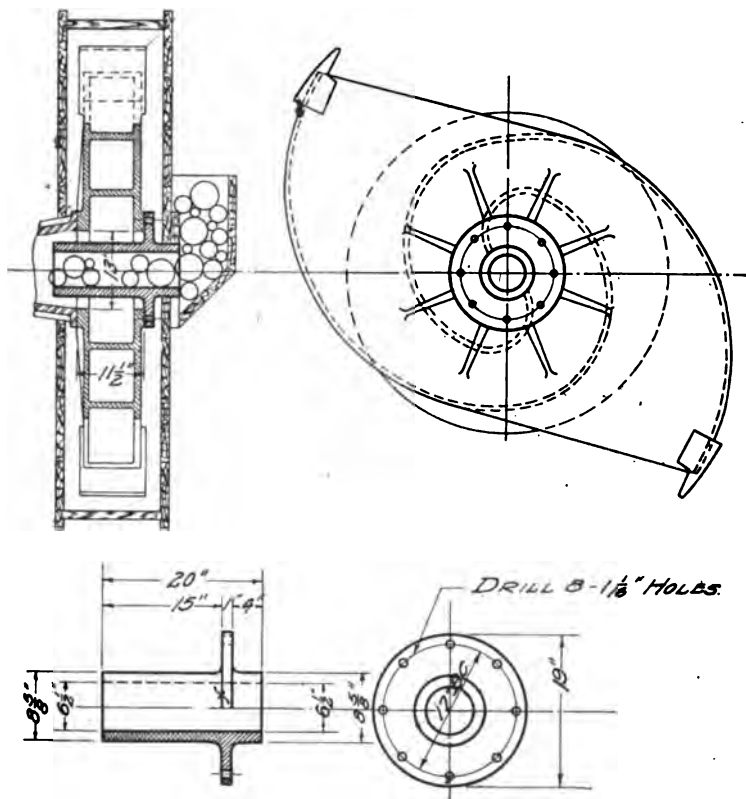


FIG. 3.

dewatering bins, together with the fine jig concentrate, and the middling is sent to the 10 by 4-ft. Hardinge mills. The overflow from the Anaconda classifiers is sent to the slime thickener division, consisting of 28 by 3-ft. Dorr tanks.³ The spigot product from these tanks is divided; about one-half is returned to the section and the remainder is sent to the slime plant.

The product from the Hardinge mills (Fig. 2) is treated in six simplex



Patent applied for.

FIG. 4.—SPOUT FOR FEEDING PEBBLES OR BALLS INTO HARDINGE OR TUBE MILLS.

Dorr classifiers—one classifier to each mill—the overflow going to the flotation division and the classifier sand being returned to the mill.

At the time it was first decided to remodel the concentrator, it was not definitely known whether pebbles or steel balls would be used for grinding. To provide for this uncertainty a compromise was effected. The mills were made 10 by 4 ft. and built sufficiently strong for steel balls in case balls were used. Each mill was equipped with a 225-hp. motor directly connected through a flexible coupling. The mill filled

³ For description of this thickener plant, see *Trans.*, xlix, p. 470 (1914).

with pebbles takes from 95 to 115 hp. to operate. In case steel balls were used it was planned to put in a false wood lining back of the steel lining in the cylindrical part of the mill to reduce the effective diameter of the mill.

This latter plan was finally adopted, and the Hardinge mills will be equipped with the false wood lining, 15 in. thick, in the cylindrical part of the mill, and a Cascade steel lining. With this form of lining, the mill is virtually $7\frac{1}{2}$ by 6 ft. and requires about 225 hp. when loaded with steel balls.

The drawing (Fig. 3) gives the details of the lining. This lining was designed by the American Manganese Steel Co.⁴ At first the pebbles, and later the balls, were fed to the mills through the feed scoop. This method of introducing the grinding medium into the mill gave considerable trouble, due to the breaking of the feed boxes caused by the jamming of a pebble or ball between the revolving scoop and the feed box. We tried to obviate this difficulty by various changes in the amount of clearance left between the scoop and the box, but without success. In our particular case this trouble was aggravated by the fact that we had to use 7 ft. diameter scoops, in order to lift back into the mill the sand discharged by the Dorr classifier. Finally a method was tried of feeding the pebbles, or balls through a spout passing through the center of the feed scoop. This device has worked splendidly and all of our mills have since been equipped with it (see Fig. 4).

FLOTATION DIVISION

The flotation division consists of four Minerals Separation machines, each having 15 agitators 3 ft. square, and 14 spitzkästen or floating compartments. The agitators for the Minerals Separation machines are of gun metal and are driven by bevel gears from a line shaft, the direction of rotation of the agitators alternating.

The machines are made of California red wood; the agitator boxes are further lined with hard maple extending about 18 in. from the bottom of the box.

Each machine has an individual drive, power being supplied to the line shaft by a 150-hp. motor running at 385 r.p.m. The speed of the agitators is 225 r.p.m. and as the impellers are 18 in. in diameter the peripheral speed is about 1,060 ft. per minute.

Each machine makes three products, a concentrate, which goes to the dewatering division, a middling which is returned to the head of the machine, and a tailing which goes to waste. The concentrate is taken from the first three to five spitzkästen and the middling from the last nine to eleven. A portion of the pulp is overflowed from the last three

⁴ A detailed description of these mills, together with grinding data and Dorr classifier data, will be published in a subsequent paper.

spitzkästen together with the froth. About 6 to 8 lb. of 50°Bé. H_2SO_4 per ton of flotation feed is used together with 2 to 3 lb. of kerosene sludge acid and $\frac{1}{2}$ to 1 lb. of crude wood creosote. A portion of the wood creosote is added ahead of the Hardinge mill (about 0.03 to 0.05 lb. per ton of feed) and the remainder is added in the sixth agitating compartment. The sulphuric acid and sludge acid are added at the head of the machine. The pulp is heated to from 60° to 70°F., by passing live steam into it at the head of the machine. Three machines are used for treating sand and the fourth for treating current slime from the upper portion of the mill. Each machine has a capacity of about 400 tons per day on sand and 175 tons on slime.

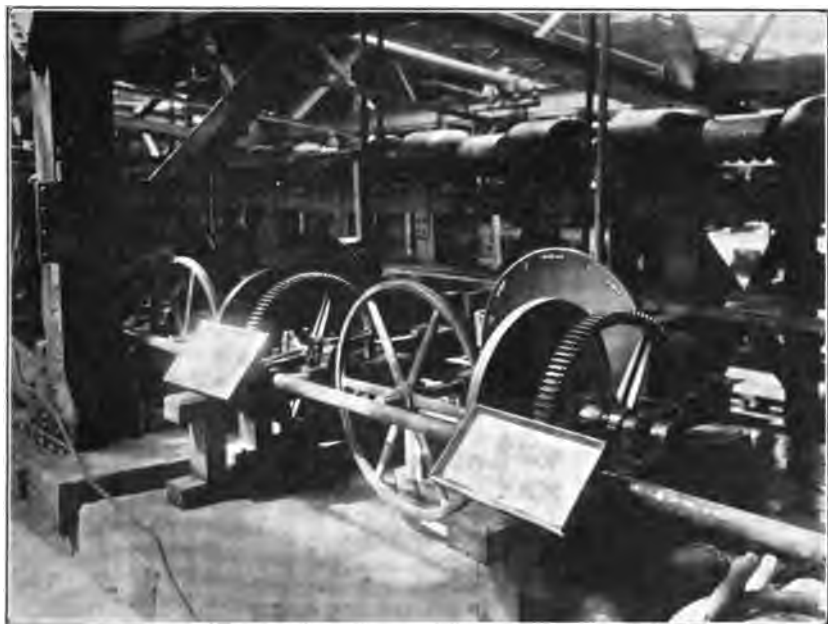


FIG. 5.—AUTOMATIC REAGENT FEEDERS.

The method of adding the oil and acid is rather unique. The mechanism consists of a revolving disk to which are attached, around the circumference, a number of cups. This disk is set vertically so that its lower edge dips into a pan of acid or oil. As the cups come around they are filled and later discharge their contents into a suitable launder leading to the flotation machine. The disk is driven by the friction of a wheel against another disk attached to the main drive. The wheel is run at constant speed and by varying the point of contact between wheel and disk any speed desired can be given to the main disk and thus the amount of oil or acid added can be regulated. In addition to the speed regulation, the amount of oil or acid fed may be varied by adding or removing cups

or by changing the size of the cups. A photograph of the machine is shown in Fig. 5.

At present (December, 1915) seven sections are operating on the new flowsheet, and the whole mill will be remodeled not later than Jan. 15, 1916. The sections are being remodeled one at a time. All the work is being done by the company's engineering force. Things have been so well organized and systematized that it requires less than 30 days to tear out the old section and install the new equipment, ready for operation.

The following table gives metallurgical data concerning the operation of the mill. Line 1 gives the monthly assay of second-class ore for October, which is the feed to the mill. Line 2 gives the monthly assay of the flotation tailing from the remodeled sections for October. The feed to the flotation machines during the month consisted of reground sand tailing and a portion of the thickened mill slime which was returned to the mill and treated in the fourth machine in each remodeled section. The tailing assay shown below, 0.13 per cent. Cu, is for the total sand and slime tailing produced. The sand tailing alone averaged 0.10 per cent. Cu and the slime tailing averaged about 0.25 per cent. Cu.

	Per Cent. Cu	Per Cent. SiO ₂	Per Cent. Al ₂ O ₃	Per Cent. Fe	Per Cent. S	Per Cent. CaO	Ounces per Ton	
							Ag	Au
Feed to concentrator.....	2.85	59.1	9.4	10.6	11.8	0.6	2.00	0.0070
Total concentrator tailing (flotation)..	0.13	81.6	11.3	1.0	0.5	0.5	0.10	0.0005

No complete analyses of the flotation concentrate are available, as the slime flotation plant is not in operation yet and the round-table concentrate is being mixed with the flotation concentrate from the mill.

Following is an estimate of the power consumption per 2,000-ton section, including its proportion of slime treatment.

	Power Consumption per Section, Horsepower
Feeding and coarse crushing.....	125
Roll crushing.....	220
Jigging.....	60
Screening.....	37
Conveying and elevating.....	237
Table concentration.....	25
Fine grinding.....	655
Flotation of sand and slime in mill.....	426
Dewatering mill slime pulp.....	3
Flotation of slime in slime division.....	109
Dewatering flotation concentrate.....	25
Sampling and assaying.....	1
Total.....	1,923
Horsepower per ton of ore.....	0.96

III. DESCRIPTION OF SLIME-FLOTATION PLANT

Because of lack of space in the mill an auxiliary plant had to be installed to handle the extra slime. This plant consists of 20 Minerals Separation machines of the same type and size as those used in the concentrator, and five 50 by 12-ft. Dorr tanks for dewatering the concentrate (Figs. 6, 7 and 8). The plant is designed to treat about 2,000 tons of current slime and 1,000 tons of pond slime. This plant will be operating by Dec. 15, 1915.

What this plant may be expected to do, so far as the treatment of current slime is concerned, may be judged from the results obtained in the experimental slime machine, which has been operating regularly for more than a year.

The following table gives the analysis of the current slime and the assay of the composite tailing sample for the month of June. These are typical of the results which may be expected in the large plant.

Operation of Experimental Slime-Flotation Plant

Per Cent.	Feed (Current Slime)	Tailing	Concentrate
Cu.....	2.10	0.27	12.0
SiO ₂	61.00	67.70	20.0
FeO.....	4.10	2.70	28.0
S.....	4.40	1.10	
Al ₂ O ₃	19.00	18.30	
CaO.....	0.60	0.70	
Ag ¹	1.80	0.20	
Au ¹	0.005	0.001	

¹ Ounce per ton.

The treatment of the pond slime offers a few more difficulties than that of the current slime. This slime has been exposed to the weather for a number of years so that the copper is partially oxidized, consequently that portion probably cannot be floated. Some results from the flotation plant at Great Falls, covering a week's operation, are tabulated below. This character of material has been treated for some time at the Great Falls plant, and the results should be a good criterion of what may be expected here in the treatment of dump slime.

Operation of Slime-Flotation Plant at Great Falls

	Per Cent.
Recovery based on concentrate.....	78.90
Recovery based on tailing.....	80.90
Average total Cu in tailing.....	0.53
Average soluble Cu in tailing.....	0.13
Average sulphide Cu in tailing.....	0.40
Average total Cu in feed.....	2.52



FIG. 6.—MINERALS SEPARATION MACHINES IN SLIME-FLOTATION PLANT.



FIG. 7.—MINERALS SEPARATION MACHINES IN SLIME-FLOTATION PLANT. (MACHINE ON LEFT IS NOT IN OPERATION.)

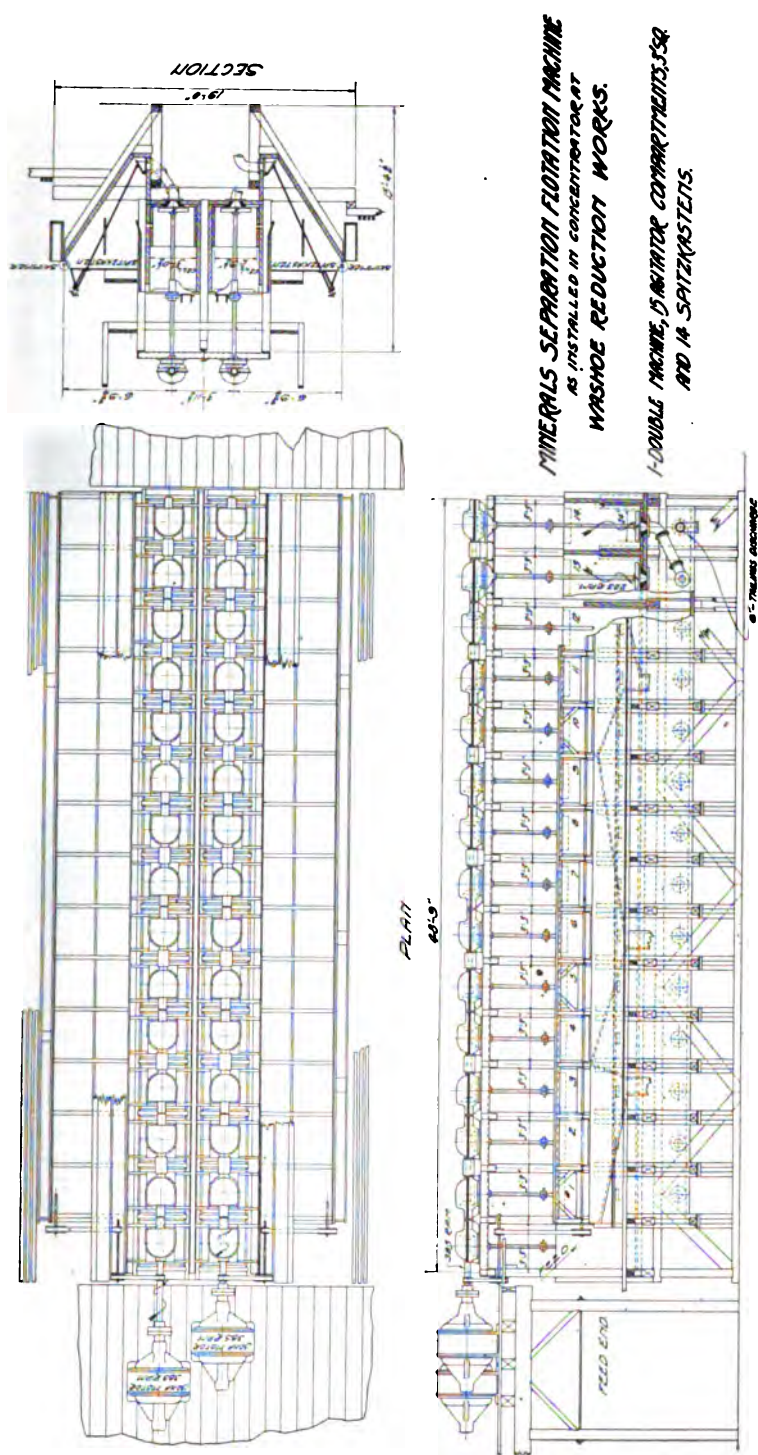


Fig. 8.

together with the dump slime. No difficulty was found in sulphidizing practically all of the copper which went into solution, but it was found that all oxidized copper was not dissolved during its passage through the pug mill and flotation machine.

Moreover, on the addition of sodium sulphide, the froth immediately broke up—a large part of the natural sulphides no longer floating, and the tailing assay showing a marked increase, although the precipitated sulphides floated. Hardly any froth was formed, probably because of the colloidal state of the precipitated sulphides rather than the result of any deleterious action on the oils used of the sodium sulphide, or the sodium sulphate formed.

The cause of the breaking of the froth was not clear. It may have



FIG. 10.—DORR TANKS IN SLIME-FLOTATION CONCENTRATE-DEWATERING DIVISION.

been due to the sodium sulphate formed, or, what seems more probable, to some impurities in the sodium sulphide, the latter being only about 50 per cent. pure. This effect was very general, the froth on the second machine breaking up, although no sodium sulphide was added to it, except through a return middling elevator which was common to both machines. Fig. 9 shows the flow sheet of the slime-flotation plant at Anaconda.

IV. DESCRIPTION OF FLOTATION CONCENTRATE DEWATERING PLANT

The dewatering of the flotation concentrate is done in six 50 by 12-ft. Dorr thickeners. Five tanks of the same size have been provided for the slime plant. The pulp is delivered to a baffle box about 5 ft.

square in the center of the tank, and extending down to within a few inches of the rake arms. Surrounding this is another baffle about 15 ft. square and extending about 18 in. below the surface of the water (Fig. 10). These baffles catch a large portion of the froth which is there broken up by means of a water spray. The capacity of these tanks when treating flotation concentrate is from 200,000 to 250,000 gal. of pulp per 24 hr., although they will probably not be required to handle more than 200,000 gal. Operating even at this capacity, there is a small amount of finely divided material that will not settle. Therefore, the overflow from these tanks is run to a slime pond and the solids collected for future treatment. The capacity of these same tanks when treating round-table concentrate is about 1,000,000 gal. of pulp per 24 hr.

Some experiments were made to increase the capacity of the tanks by the use of gluc. Indications from tests in a beaker seemed to show that glue would greatly aid the settling, but this did not prove to be the case. The glue caused the colloidal material to coagulate, but it did not actually increase the capacity of the settling tanks to any great extent.

The density of the pulp delivered to the tanks averages from 18 to 20 per cent. solids, and the spigot averages about 60 per cent. solids. The spigot product is further dewatered in Oliver filters 11½ ft. diameter by 12 ft. face. There will be eleven of these, each capable of handling 150 tons per 24 hr. and making a cake containing about 15 per cent. moisture. The cake from the filters is delivered to a belt conveyor and is carried with the fine mill concentrate to the new roaster plant.⁵ In this way a fairly good mixing of the concentrates will be obtained.

⁵ The new roaster plant consists of 28 25-ft. Wedge furnaces. This will about double the roaster capacity of the plant.

TABLE I.—Summary of Flotation Tests—Minerals Separation Machine No. 1
Treatment of Slime: Round-Table Feed and Tasting

Period	Feed		Concentrate Assay				Tailing Assay, Per Cent. Cu	Per Cent. Cu Recovered	Temperature, °F.			Reagents Used, Pounds per Ton					P C A T		
	Tons per 24 Hr.	Assay, Per Cent. Cu	Density, Per Cent. Solids	Hours in Opera- tion	Per Cent. Cu	Per Cent. Insol.			Per Cent. FeO	Ag Oz. per Ton	Feed Pulp Machine	Pulp in Difference	Sludge Acid	Wood Creo- sote	Stove Oil	H ₂ SO ₄		Tur- pentine	Tar Creo- sote
1 Preliminary																			
2	61.9	2.04	13.7	120.0	8.2	54.7	13.7	6.0	0.31	89.3	54	89	3.8	2.4	0.36	12.6			1.61
3	64.2	2.07	14.9	72.0	7.2	62.0	10.8	6.0	0.38	86.0	51	51	3.6	2.3	0.29	13.3			1.73b
4	56.2	2.08	13.6	48.0	8.3	51.1	15.6	6.7	0.27	90.4	53	125	3.9	2.8	0.33	12.6			3.60
5	52.2	1.16	11.7	56.0	9.6	66.1	7.8	5.8	0.33	75.5	54	54	3.5	3.8	0.37	15.2			1.88
6	48.2	1.08	11.1	42.0	7.2	62.1	8.9	6.8	0.31	74.3	54	106	3.9	4.2	0.29	11.8			1.52
7	63.8	2.24	22.5	72.0	9.2	52.9	12.8	7.8	0.28	90.3	53	96	4.3	2.4	0.33	12.5			1.38
8	104.8	2.19	24.5	88.0	8.9	57.5	11.2	6.45	0.63	76.6	53	96	4.3	2.4	0.32	13.9a			1.52
9	94.7	2.25	22.7	120.0	8.7	60.5	9.2	6.45	1.34	47.5	54	54	2.0	2.2	0.21	16.4			1.52
10	60.4	2.29	14.5	96.0	10.1	52.6	12.4	7.9	0.50	82.0	48	48	3.3	2.9	0.42	12.7			1.52
11	58.2	2.39	14.5	144.0	9.2	53.3	12.8	6.8	0.33	89.5	50	81	3.1	3.5	0.21	16.5			1.52
12	56.1	1.11	18.1	24.0	5.9	71.8	5.2	4.4	0.53	57.8	52	72	20	3.7	0.28	16.1			1.52
13A	60.6	1.11	13.5	184.0	5.8	71.3	5.6	4.2	0.28	79.1	52	80	28	3.7	0.30	16.8			1.52
13B	51.6	1.35	11.1	32.0	2.0	85.0	2.5	1.0	1.04	47.8	51	80	29	5.5	0.27	00.0			1.52
14A	31.7	1.17	5.6	48.0	6.8	66.2	7.4	4.6	0.30	77.8	50	80	30	5.1	0.32	18.9	3.7		1.52
14B	42.6	1.17	6.8	24.0	5.9	68.0	7.2	4.6	0.24	82.7	50	71	21	4.4	0.37	21.0			1.52
14C	40.4	1.12	6.7	80.0	6.1	67.2	8.3	4.8	0.23	82.5	52	80	28	4.4	0.28	18.5			1.52
14D	40.0	1.13	6.5	16.0	9.8	54.3	11.5	6.6	0.32	92.5	52	81	29	5.0	0.33	20.1			1.52
15A	42.7	2.35	9.4	40.0	9.3	47.2	17.0	6.7	0.23	91.5	48	82	34	4.3	0.33	20.7			1.52
15B	48.0	2.51	8.0	32.0	13.4	33.6	20.9	8.8	0.26	91.5	48	78	36	3.8	0.41	19.3	4.7		1.52
15C	49.0	2.52	8.8	80.0	10.8	44.2	18.4	8.3	0.25	92.2	45	71	26	3.4	0.34	20.0			1.52
15D	49.4	2.25	9.7	40.0	11.0	40.7	19.3	8.1	0.38	86.0	46	70	24	9.0	0.33	19.3	3.3		1.52
15E	50.2	2.52	9.3	40.0	14.3	28.9	23.6	11.4	0.48	83.8	44	70	26	4.0	0.49	18.6			1.52
16A	49.0	2.55	11.5	248.0	9.7	48.5	16.0	7.7	0.27	92.1	44	68	24	3.2	0.47	20.9			1.52
16B	55.2	2.58	13.7	24.0	10.6	48.5	15.6	9.2	0.56	83.0	45	66	21	5.9	0.46	20.9			1.52
17	40.0	1.10	5.8	576.0	7.5	63.0	9.3	5.9	0.22	82.4	43	69	26	4.7	0.31	23.2			1.52
18	42.0	1.11	5.8	334.9	7.0	63.9	7.7	4.7	0.26	79.5	40	70	30	4.1	0.30	16.7			1.52
19	90.0	2.57	15.7	236.4	14.8	31.3	22.1	11.2	0.43	85.8	39	72	33	3.27	0.26	13.1			1.52
20	60.2	2.74	15.3	211.3	12.0	38.3	20.6	8.8	0.33	90.5	39	70	31	3.27	0.26	13.1			1.52
21	40.9	2.48	15.1	313.1	10.1	43.7	18.8	6.9	0.25	92.3	41	74	33	3.27	0.29	19.5			1.52
22	88.2	2.54	15.9	417.7	11.9	39.4	20.1	8.4	0.25	92.3	41	74	33	3.27	0.13	13.5			1.52
23	89.9	2.60	15.4	283.7	12.3	36.4	20.8	9.3	0.27	91.6	43	74	31	3.27	0.18	14.9			1.52
30c	93.3	2.55	10.7	336.0	8.3	55.2	13.3	5.8	0.41	99.3	44	65	21	3.7	0.71	21.1			1.52
31c	45.1	1.10	5.7	720.0	4.5	74.3	5.7		0.27	80.3	44	65	21	3.9	0.34	18.7			1.52

NOTE.—This table gives the average of the results of each period.

a During this period about 1.8 lb. of fuel oil per ton of feed was used with the other oils.

b During two days of this test Midwest Refining Co. sludge acid was used along with the other reagents at the rate of 3.69 lb. per ton, and on another day Western Oil Co. sludge acid at the rate of 1.04 lb. per ton. No improvement found in either instance.

c These tests were made in M. S. Machine No. 2.

TABLE IA.—Summary of Tests in Minerals Separation Machine No. 1.—Continued
Treatment of Sand Tailing from Mill, Ground through 0.25 Mm.

Period	Feed		Concentrate Assay				Tailing Assay, Per Cent. Cu	Per Cent. Cu Recovered	Temperature, °F.			Reagents Used, Pounds per Ton					L. P. O. 400
	Tons per 24 Hr.	Assay, per Cent. Cu	Density, Per Cent. Solids	Hours in Operation	Per Cent. Cu	Per Cent. Insol.	Per Cent. FeO	Per Cent. Ag	Feed Pulp Machine	Dif- ference,	Sludge, Acid	Wood Creso- sote	Stove Oil	H ₂ SO ₄	Tur- pentine M. S. 14	Tar Creso- sote	
24 Preliminary																	
25	125.0	0.62	22.3	55.8	6.35	40.1	26.8	4.2	40	29	2.5			7.6			
26	186.7	0.87	30.9	472.6	7.82	26.9	32.5	5.4	43	26	3.1	0.7		4.9			0.3 ¹
27	124.1	0.67	26.1	120.5	5.42	42.2	25.4	4.2	42	30			0.3	1.3	3.9		

¹ Average of four days' test. Results not improved by this oil. For two days 1/4 lb. of Standard Oil Co. Sludge Acid Kerosene per ton of feed was used along with the P. T. & T. Co. 400. No marked improvement noted. Wood creosote used for only six days. Tailing assayed about 0.09 during this time.

TABLE II.—Treatment of Round-Table Feed by Flotation Process. Minerals Separation Machine No. 2A

Date, 1915	Rate per 24 Hr.		Assay Feed, Per Cent. Cu	Temperature of Circuit, °F.	Tailing Assay, Per Cent. Cu	Concentrate, Per Cent.			Reagents, Lb. per Ton			
	Gallons Pulp	Tons Solids				Cu	Insoluble	FeO	H ₂ SO ₄	M. S. 37	P. T. & T. Co. 400	M. S. 8
2-27	58,900	39.5	2.59	90.9	0.86	8.3	54.4	14.3	26.3	6.9	2.3	0.15
2-28	40,200	29.8	2.50	87.2	0.73	9.8	36.9	27.1	30.1	7.0	3.2	0.04
3-1	44,600	29.7	2.50	86.1	0.93	10.0	46.4	18.3	26.1	7.3	5.3	0.50
3-2	49,300	34.6	2.55	92.6	0.50	5.9	62.2	11.9	20.9	6.5	2.7	0.20
3-3	52,200	36.4	2.62	90.7	0.47	6.0	60.5	12.6	22.3	6.1	1.3	0.07
3-4	55,100	39.7	2.54	86.0	0.61	7.0	58.0	13.0	17.6	5.1	2.4	0.03
Average.....	50,050	34.9	2.55	88.9	0.68	7.8	53.1	16.2	20.5	6.5	2.9	0.16

The machine was in good running order, mechanically, on Feb. 27, 1915. There seemed to be too much agitation at first, so No. 4 impeller was dropped off shaft. Machine ran this way until Mar. 3, when the No. 2 impeller was dropped. Air was blown in under each impeller. With first and third impellers operating, the input to the motor was 32 hp.

TABLE III.—Summary of Flotation Tests—Callow Pneumatic Machine
Treatment of Slime

Period	Hours in Operation	Number of Rougher Cells	Air Pressure, Pounds per Square Inch	Feed					Temperature of Pulp in Circulation	Reagents Used, Lb. per Ton						Final Tailing, Per Cent. Cu	Final Concentrate			Mid- dling, Per Cent. Cu	Rougher Concentrate			Per Cent. Cu Recovered																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																	
				Per Cent. on 80 Mesh	Density, Per Cent.	Solids	Assay, Per Cent. Cu	Tons per Rougher Cell		H ₂ SO ₄	Sludge Acid	Crude Wood Creosote	Stove Oil	Turpentine	Fuel Oil		Pine Tar	Per Cent. Cu	Per Cent. Insol.		Per Cent. FeO	Per Cent. Cu	Per Cent. Insol.		Per Cent. FeO																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																																
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a Air agitation used throughout this period.

b These oils were used only during first five days.

c Mechanical agitation used throughout.

e Concentrate assayed 5.2 oz. silver per ton.

d Concentrate assayed 4.8 oz. silver per ton.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co.

BY H. KENTON BURCH,* B. S., MIAMI, ARIZ., AND M. A. WHITING,† SCHENECTADY, N. Y.

(Arizona Meeting, September, 1916)

ONE of the advantages presented by electric drive in many classes of work is the ease with which the electric motor can be controlled automatically. In a large number of cases certain features of the control are automatic—for example, the rate of acceleration may be limited automatically or the equipment may be stopped automatically at the limit of travel—but the equipment is started and ordinarily is stopped by hand. In other cases the motion of the machinery is utilized to start, control the speed, and stop the motor automatically, independently of any operator.

A considerable proportion of the large mine hoists now in use have certain automatic features, particularly protective devices against overwinding, and, in some classes of electric hoists, devices for preventing excessive acceleration or retardation. The large automatic hoists discussed in this paper, however, are completely automatic, *i.e.*, capable of making their trips without the presence of an operator at the control levers.

According to circumstances, various advantages may be obtained by automatic control, chief of which are decreased power consumption, increased precision and safety of operation, and decreased cost of attendance. The first step in the analysis of a prospective automatic mine hoist is to determine whether automatic operation is feasible at all. If men are to be hoisted, or levels changed, the attention of an operator is required for these purposes; but under some conditions it may be entirely practicable and advantageous to build the equipment so that, while provision is made for hoisting men or changing levels, ore can be hoisted automatically from any one level. If, however, an operator's attention is required every few minutes for changing levels, handling men or drills, or other work requiring hand control, it is obvious that automatic operation between times will not be of any practical benefit.

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† Engineer with Power & Mining Department, General Electric Co.

SPEED CONDITIONS REQUIRED FOR AUTOMATIC HOISTING

For a very slow hoisting speed it may be possible for the skip or cage to pass through the dump at full speed, and a sufficiently accurate stop may possibly be obtained automatically by cutting off power and applying the brakes at full speed. In this case, either a shunt-wound direct-current motor or an induction motor may be used. A number of slow-speed automatic hoists are arranged in this manner and are driven by induction motors. One equipment of this type used in mining work is the inclined hoist for handling concentrates at one of the mills of the Arizona Copper Co., described by H. L. Hall in the *Mining and Engineering World* for Apr. 10, 1915. This hoist has a rope speed of approximately 275 ft. per minute.

For higher rope speeds, at least for speeds over 400 ft. per minute, it is necessary to consider carefully the speed characteristics obtainable from the type of drive proposed. For these higher rope speeds, it is necessary to slow down before entering the dumping horns. Furthermore, the speed about midway in the dump must usually be reduced below the maximum safe speed entering the dump. A reasonably accurate stop is always required; in some cases a total variation of 2 or 3 ft. might not prove prohibitive, but in other cases the stop must be more accurate. For reliable operation, it is nearly always imperative that the automatic-control system shall act in like manner irrespective of load, i.e., that the rate of retardation and the position of stopping be nearly the same whether the skip comes up loaded or empty.

There is only one class of motive power which is inherently suited for automatic operation at high rope speeds, viz., the direct-current shunt-wound motor with voltage control. The speed-torque characteristics for an equipment of this character are represented in Fig. 1. These curves are typical of this class of equipment, although the exact slope of the curves will vary slightly in individual cases. Curve 1 shows the characteristics on the lowest, and curve 5 the characteristic on the highest, speed position of the controller for the case selected. The intermediate curves represent three controller points arbitrarily selected out of a total of 30 or more. It will be observed that these curves are nearly, but not quite, parallel. That is to say, the increase in speed in passing from full load to no load is approximately, but not exactly, the same for the various positions of the controller. The deviation from parallelism is due to the effect of armature reactions in the generator and hoist motor, and may be somewhat different for different cases; but its effect is negligible.

The net advantages (for the purpose of automatic hoisting) obtained by this system of drive are as follows:

As the hoist controller is moved back toward the off position the

hoist is retarded. In case the net rope pull is sufficient and the stored energy of the moving system is not too great, the hoist motor simply drops back in speed to correspond to the reduced generator voltage obtained on the intermediate position of the controller. If, however, the net rope pull is very low (particularly with empty skips in balance), and if the stored energy of the moving system is high, the hoist motor will invert, momentarily, and will act as a generator, returning power to the motor-generator set. This effect is represented in Fig. 1 by the extension of the curves below zero torque. In this manner, if the controller

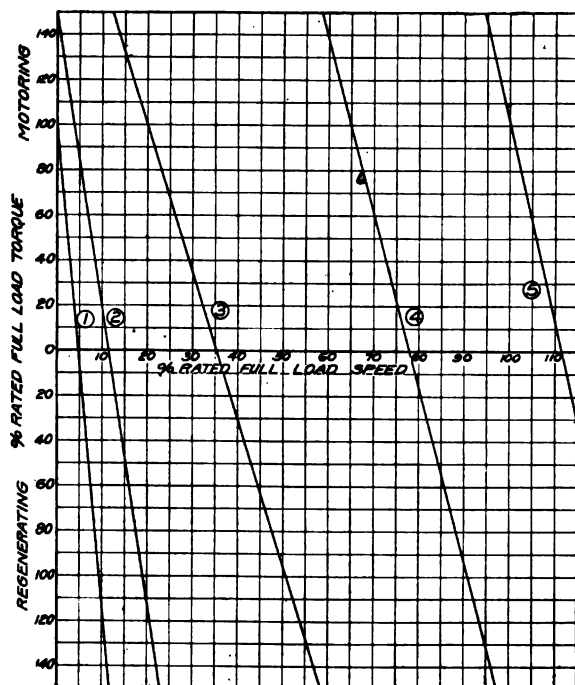


FIG. 1.—TYPICAL SPEED-TORQUE CURVES FOR DIRECT-CURRENT MINE HOIST WITH GENERATOR FIELD CONTROL.

is moved toward the off position more rapidly than the hoist tends to come to rest under the influence of the load, the hoist motor forcibly retards the hoist. If the controller is moved back at the same rate in both cases, the hoist will be retarded to nearly the same speed, and in nearly the same time, irrespective of load in the skip.

It is fairly obvious that the steam hoist is unable to approach very closely the speed conditions just described. The steam hoist, of course, is capable of retarding a load by working against the steam or compression, but the vital points in relation to automatic hoisting are: (1) for the same throttle opening and cutoff, the speed will vary widely with

variation in load; and (2) if the throttle is partly closed or the cutoff advanced to a point at which the skip will enter the dump at a suitable reduced speed, the engine will exert only a slight retarding torque (if any) to help retard from full speed to the reduced speed at which the engine tends to continue. Most of the retardation must therefore come from the load, which is variable, or may even be negative. Furthermore, with a partly closed throttle the final speed, at which the engine tends to continue, will vary widely with variation in load.

The induction-motor hoist, in its relation to automatic hoisting, has somewhat the same characteristics as the steam hoist. Fig. 2 represents the speed-torque characteristics of a typical mine-hoist induction motor. In a direct-current hoist, a given retardation can be accomplished in a certain time and distance by the same manipulation of the control, irrespective of the load hoisted. In a steam or air hoist or an

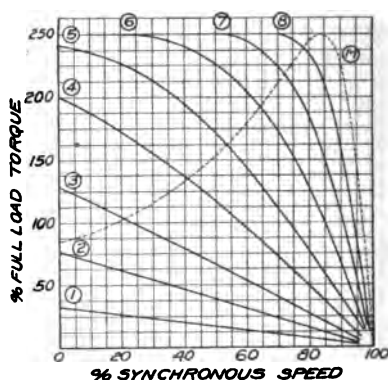


FIG. 2.—TYPICAL SPEED-TORQUE CURVES FOR INDUCTION-MOTOR-DRIVEN MINE HOIST.

induction-motor hoist, a like retardation of different loads requires different manipulation of the control.

These characteristics indicate, and their further consideration confirms, the conclusion that high-speed mine hoists which are to be operated automatically must be, in almost all cases, driven by direct current.

THE AUTOMATIC HOISTS OF THE INSPIRATION CONSOLIDATED COPPER CO.

When the layout of their main shafts was under consideration by the Inspiration Consolidated Copper Co. a concurrence of several conditions indicated the possibility of effecting a saving by hoisting the ore automatically. These conditions were as follows: (1) A direct-current equipment was necessary in any case, as a motor-generator set was required for the flywheel equalization as provided in the power contract

with the Reclamation Service. (2) The ore was all to be hoisted from one level. (3) Drills, timbers, supplies and waste were to be handled through a drift opening. (4) Men were to be handled on a separate hoist exclusively. (5) On account of the moderate depth and rope speed only a moderate retardation effort would be required.

General Arrangement of Hoists

Two three-compartment shafts have been sunk, for two independent balanced hoists, each hoist being adequate in an emergency to keep the concentrator operating at practically full capacity. The third compartment of one shaft contains a double-deck man cage, operating against

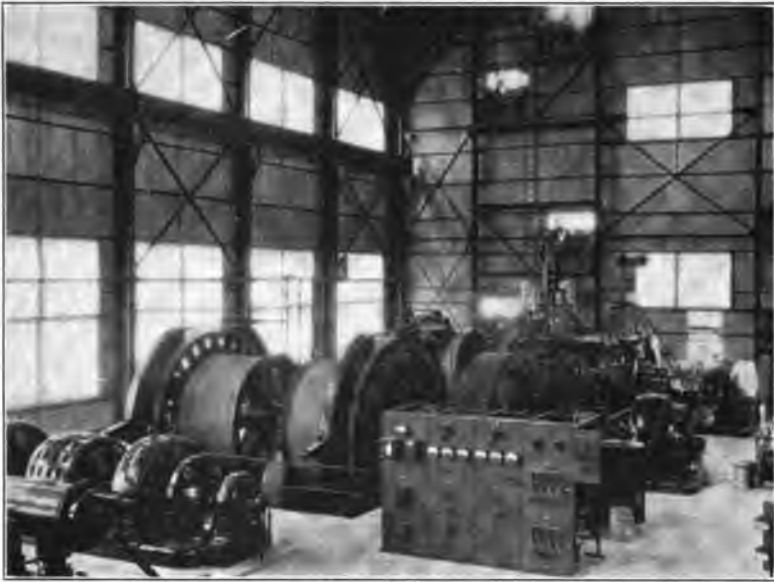


FIG. 3.—MAIN HOISTS, INSPIRATION CONSOLIDATED COPPER CO.

a counterbalance weight, and the third compartment in the other shaft carries this counterweight, together with air lines, electrical cables, etc. Skips carrying $12\frac{1}{2}$ tons are used, and the ordinary hoisting schedule for which the equipment was designed called for an output of 10,000 tons, with a maximum capacity of 14,000 tons, in 14 hr. The hoists are located in one end of the compressor house. No. 2 hoist, in the background in Fig. 3, handles the skips in the East shaft, which is nearest the compressor house. No. 1 hoist, in the foreground, handles the skips in the West shaft, the ropes from No. 1 passing above No. 2 hoist over idler sheaves on the upper deck of the East headframe, thence over the sheaves on the West headframe. Fig. 4 shows the arrangement of shafts and headframes in relation to the compressor house.

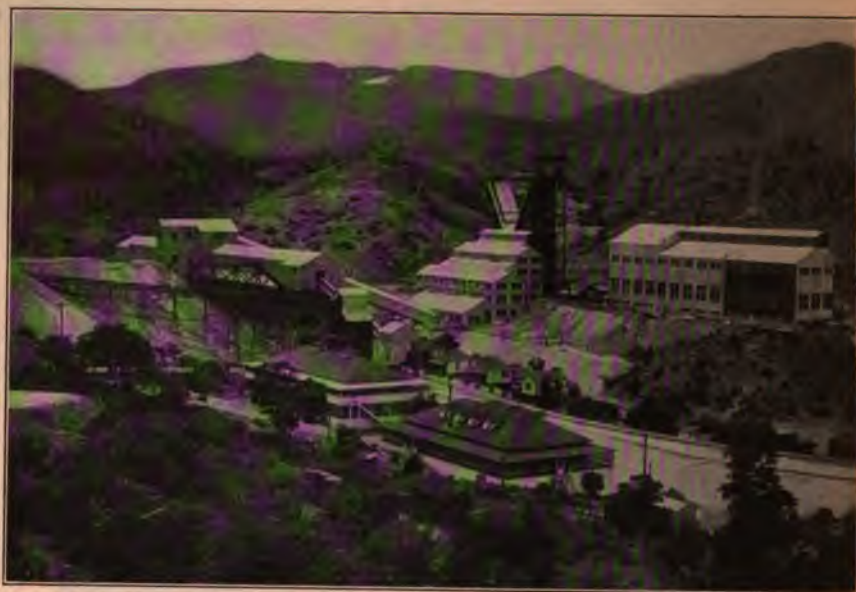


FIG. 4.—MAIN SHAFTS, COMPRESSOR HOUSE, COARSE-CUSHING PLANT AND STORAGE BINS, INSPIRATION CONSOLIDATED COPPER CO., MIAMI, ARIZ., DURING CONSTRUCTION PERIOD.

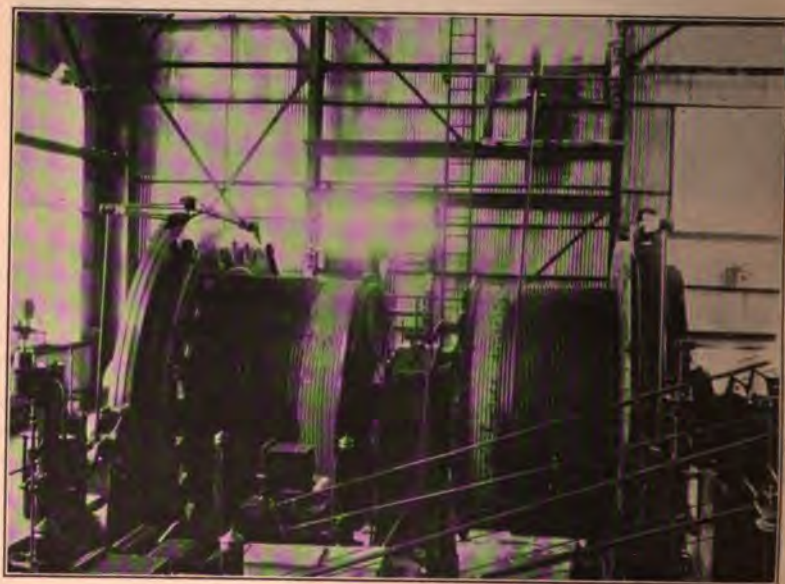


FIG. 5.—DRUMS AND BRAKES OF ONE MAIN HOIST.

The hoists are duplicates, each consisting of one fixed and one clutched drum, each 10 ft. diameter by 65-in. face, grooved for 1,000 ft. of 1¾-in. rope in one layer. The brakes and clutches are air operated with oil cataracts and floating levers, and the automatic control system was so designed that the brake engines could be made practically standard (Fig. 5.) The hoists were designed and built by the Nordberg Manufacturing Co. and the electrical equipment by the General Electric Co.

Each hoist is driven by a 580-hp., 575-volt, 264-r.p.m. shunt-wound motor through a flexible coupling and Falk gears. Power is supplied to the hoists by a 750-r.p.m. flywheel motor-generator set, consisting of one 850-hp., 2,300-volt, 25-cycle induction motor, two 500-kw., 575-



FIG. 6.—LIQUID SLIP REGULATOR FOR REGULATION OF INPUT OF FLYWHEEL SET.

volt generators, one 20-kw., 125-volt exciter and a 19,700-lb. 112-in. diameter steel-plate flywheel. Each hoist motor is connected separately to one of the generators and controlled by varying the field of its generator. The flywheel is not in any way necessary to the control or automatic operation of the hoists. Its function is to eliminate the peak-loads from the power system. The control for equalization of the power demand follows along standard lines, using a liquid slip-regulator for varying the speed of the flywheel set by varying the resistance in the secondary circuit of the induction motor (Fig. 6).

The depth, from the dump to the chairs under the loading pockets, is 630 ft. in each shaft; from the collar to the chairs, 557 ft. The rope speed is approximately 750 ft. per minute.

Description of Automatic Cycle

Before beginning automatic operation it is necessary, of course, that each hoist be properly clutched-in for the loading level, with one skip in each shaft resting on the chairs below its loading chute. It is not important which skips are on the chairs, provided, of course, that the operator obtains a "release" of skips in both shafts before starting the automatic operation. He then introduces the automatic control by closing two small control switches and locking in two levers, all on the operating-platform. This does not, of itself, start the automatic operation, so that the hoists may be left standing in this manner indefinitely. To start the automatic operation, a master controller is thrown to the automatic running position, and left there as long as automatic hoisting continues. According to the positions in which the skips have been resting, one hoist or the other will start. Say, for example, No. 1 hoist starts, hoisting its South skip. The closing of the master controller just mentioned energizes a small pilot motor which moves No. 1 hoist controller gradually to the full-speed position in one direction. As No. 1 controller starts away from the off position, it simultaneously energizes No. 1 generator field and actuates a pilot device which releases the brakes on No. 1 hoist. As the controller moves farther toward the full-speed position, it gradually builds up the generator voltage, thereby accelerating the hoist to full speed.

Toward the end of its trip the travel of No. 1 hoist actuates a pilot motor which moves No. 2 hoist controller gradually to the full-speed position in one direction, thereby accelerating No. 2 hoist in a similar manner, to hoist its North skip. Shortly before its skip enters the dumping horns, the travel of No. 1 hoist, by means of cams, one of which is geared to each drum, moves No. 1 controller gradually toward the off position. This gradually decreases No. 1 generator voltage, thereby retarding No. 1 hoist, and just as its North skip is about to land on the chairs, No. 1 controller comes into the off position. This completes the retardation and automatically applies the brakes. No. 1 hoist stands at rest while No. 2 is hoisting its North skip. Toward the end of its trip, No. 2 hoist energizes the pilot motor for No. 1 controller so as to start No. 1 hoist in the opposite direction, *i.e.*, to hoist its North skip. No. 2 hoist comes to rest in the manner described for No. 1, and rests while No. 1 is hoisting its North skip. Toward the end of its trip, No. 1 hoist energizes the pilot control to start No. 2 in the opposite direction, *i.e.*, to hoist its South skip. The sequence continues in this manner until stopped by the operator, as described later.

A loading system is used underground by which the skips are automatically loaded with a predetermined weight of ore per trip. The reduction of the attendance required at the foot of the shaft contributes

materially to the advantages of automatic hoisting. The automatic loading system can be thrown out of engagement in either shaft so that the hoists may be operated either automatically or by hand, for purposes of inspection or adjustment, without hoisting any ore.

Variation in Rate of Automatic Hoisting

To obtain a more rapid operation of the hoists, *i.e.*, a greater number of trips per hour, when operating automatically a control switch may be thrown, by which each hoist will be started earlier in the trip of the other hoist, thus overlapping to a greater extent the trips of the two. If it is desired to run the hoists automatically at fewer trips per hour than normal, this is done by introducing resistance permanently in each generator field circuit, to give a rope speed lower than normal.

Hand Control

When the details of design were first considered, one of the chief problems was the arrangement of the control so that the transition from hand to automatic operation, and more especially the transition from automatic to hand operation, might be made without risk or delay, and in a manner easily remembered by any operator acquainted with the equipment. To this end the levers on the operating platform which operate the hoist controllers and brakes for hand control are not disconnected from the controllers or brake engines when running automatically. Consequently, when the automatic pilot devices are cut in, and the hoists are operating automatically, these levers move back and forth, as if the hoists were being controlled by hand by invisible operators. When, therefore, the transition from automatic to hand operation is made during a trip, the brake and controller levers of both hoists are in the correct positions and properly in engagement for hand control.

The automatic operation can be interrupted at any time during a trip. This is done most easily by throwing the master controller for automatic operation to the off position, which causes any trip which is under way at the time to be completed automatically, dumping in the usual manner, but prevents the next trip from starting. If the hoists are then left standing, and not operated by hand, all that is necessary to start automatic hoisting again is to throw the master controller to the automatic running position (Fig. 7).

Before the construction work at the foot of the shafts and in the bins in the tippie had been completed in all details, it was necessary occasionally to stop an automatic trip without letting it dump. In such an event, or when necessary for any reason to transfer to hand control before completing a trip, the master controller for automatic hoisting is thrown

to the off position. Without disconnecting or unhooking any other parts the controller lever of the hoist which is running may then be pulled back to the off position by hand, and as the controller comes into the off position the brakes will set automatically. It is now possible to leave the pilot control of the brakes connected in service, so that the brakes will release and set automatically, as the controller is moved by hand from or to the off position. Or, if necessary on account of the character of hoisting to be done, the automatic pilot control of the brakes can be cut out, in which case brakes and controller will be controlled separately by hand.

Under all conditions (except when making adjustments in the manner

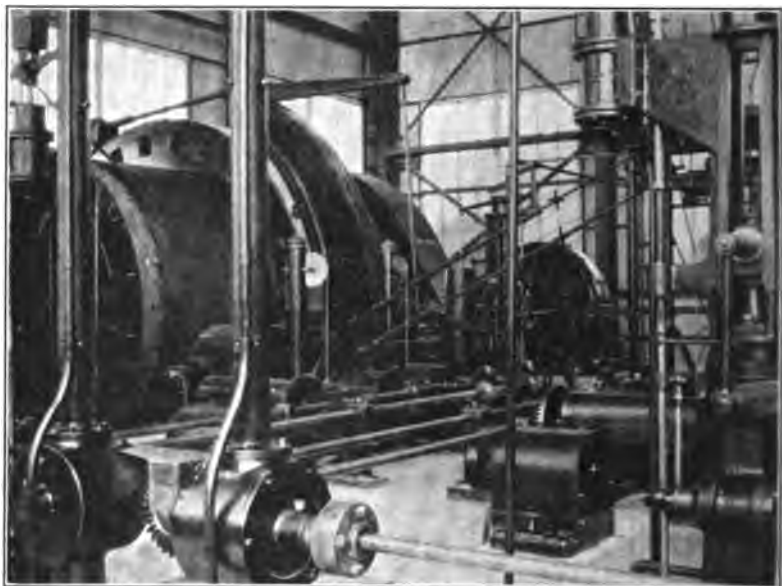


FIG. 7.—DEPTH INDICATORS AND AUTOMATIC-CONTROL SYSTEM FOR MAIN HOISTS.

described later), the cams on each hoist controller remain connected mechanically to the hoist drums. This cam mechanism thus serves two purposes: (1) in automatic operation it provides the automatic slow-down and stop; and (2) in hand operation, if the operator does not begin retardation at the proper point, this mechanism will retard the hoist in practically the same manner as when hoisting automatically, thus providing protection against overwinding when operating by hand.

Protective Devices

The protective system resembles those of a considerable number of large direct-current mine hoists, of the same general type (except the

automatic operation) as the Inspiration hoists. In the latter, as has just been noted, the automatic control system provides against overwinding in hand operation. An additional set of emergency-limit switches is used, which gives similar protection in case of failure of the automatic control. During hand operation there are effective, therefore, two complete sets of protective devices against overwinding. For each hoist a hand-operated emergency switch is provided on the operating platform, and a similar emergency switch is located at the foot of the corresponding shaft, by means of which either or both hoists may be stopped quickly from the operating platform, or the foot. Without appreciable complication, additional emergency switches may be installed at other points, if desired.

The operation of any one or more of these emergency devices cuts off power from the hoist and makes an emergency application of the brakes. An emergency, which affects one hoist only, acts on the power and brakes of that hoist only. The failure of excitation or alternating-current power, which affects both hoists, cuts off power and makes an emergency application of the brakes simultaneously on both hoists.

Adjustments in Service

When unclutching for changing levels, and when taking up stretch of ropes, the adjustments are taken care of as follows:

On the Inspiration hoists it has been the custom, whenever the hoists are to be idle an entire shift, to bring both skips to the collar of the shaft, in order to save rusting of the ropes. This is done by unclutching just as in any ordinary hoist with one fixed and one clutched drum. If desirable for any reason, either hoist may be run by hand control either out of balance or clutched in for balance to operate from other levels than the regular loading level. When clutching or unclutching, the adjustments of the automatic control system are not touched.

If the shafts are sunk to the ultimate depth contemplated, and the present loading stations abandoned, the control can be readjusted to operate automatically from the increased depth. Without changing the adjustment of the control equipment, it is not possible to operate automatically from levels differing considerably from the normal level for which adjustment has been made, but the system is capable of modification so as to hoist automatically in balance from any level to the dump, without readjustment, all the adjustments being taken care of automatically by clutching in at the desired level.

Stretch of ropes is taken up in a simple manner which itself is semi-automatic and does not require any measurements. The first time it was necessary, the stretch was taken up on both ropes of one hoist in about 15 min., at the end of which all adjustments were in shape for hand or automatic operation. The method is as follows:

The hoist is run into an automatic stop with the skip on the clutched side resting on the chairs. (This is effected by the cam which is geared to the clutched drum.) The controller and cams are now in the proper position for an automatic stop on this side; but the rope on this side has unwound farther than normal by an amount equal to the stretch or slack which it is intended to take up. This cam is now uncoupled, but the other cam is left coupled. The hoist is now moved by hand control just far enough to wind up the estimated amount of slack, and the cam on this side is then coupled up to the clutched drum. This operation takes up the slack on the clutched side and transfers it to the fixed side. The hoist is now run, in balance, into an automatic stop on the fixed drum side, which lands the skip on the fixed drum side on the chairs, and brings the skip on the clutched side into the dump. The cam on the fixed side is now uncoupled, and before moving the hoist to take up slack, the other drum is unclutched, so as to leave its skip in the normal position in the dump. The fixed drum is then moved sufficiently to take up all the slack on that side, *i.e.*, the stretch of rope on that side plus the slack transferred to that side by taking up the stretch on the clutched drum side just previously. The cam is then coupled up to the fixed drum and the other drum is clutched-in, which completes the adjustment of both ropes and cams and leaves the hoist ready for operation. It is necessary, of course, not only to take up stretch on each side, but also to clutch-in at the proper level. During the foregoing procedure, after unclutching one drum as described, the same movement of the other drum which takes up the slack also makes the necessary correction for level.

General Observations on Operation

The East shaft was ready for operation before the construction work had been completed in the West shaft. The ropes were put on No. 2 hoist, and for purposes of test and for a thorough tryout of the system, both hoists were operated automatically, No. 1 hoist running automatically as if in actual service, but without any ropes on the drums.

Both ropes were on No. 2 hoist and both skips were hung in the East shaft by the morning of July 25, 1915. During one shift on that day, after marking the ropes and the drum flanges, we coupled up the automatic control and the depth indicators to the drums, checked up the shaft and tippie clearances, and the adjustment of the cams for automatic retardation and stop, and hoisted 18 skips of ore by hand control, using the cams for automatic retardation but not using complete automatic operation. The following day, between 8 a.m. and noon, we made adjustments for complete automatic operation, and hoisted automatically 44 loaded skips. The adjustments were refined somewhat at a later date, but those made during the first three-quarters of an hour of automatic operation worked well.

The same morning in which the equipment first operated automatically, the accuracy of stop was observed for 12 consecutive trips, i.e., six trips each way. The total variation between maximum and minimum was 4 in. of rope travel. After a few weeks of intermittent operation, similar observations were taken. In 20 consecutive trips (10 each way), the total variation between maximum and minimum was only 1.5 in. of rope travel in one direction and 1.25 in. in the other. During this time the ore hung back in the loading pockets on one side, so that six of the trips included in the above figures were made empty. It is significant that this variation of 1.5 in. is only 1 per cent. of the distance traveled per second at full speed of the hoist.

Attendance

To operate two hand-controlled hoists, either steam or electric, of the size and importance of these, would require at least two operators per shift; and according to practice in some localities, an oiler would be employed in addition to the two operators.

For the operation of these two automatic hoists there is required only one operator, who is able to attend to the oiling and to whatever hand operation of either hoist may be necessary on his shift.

GENERAL CONCLUSIONS

In the section of the paper dealing particularly with the Inspiration hoists, it is our purpose principally to describe the operating features and the results accomplished by the system, as it would expand this paper to an excessive length to describe fully the essential details of design of the control equipment by which these results were attained. It would enlarge the paper still more to enter into the reasons in accordance with which the methods were selected for performing the several necessary functions of this control system. When the design of the equipment was undertaken several engineers worked, at first independently and then in consultation, in order to give due consideration to all practicable methods of operation of the various details, and as a result several arrangements were studied and discarded before settling on the one finally adopted. It was this thorough preliminary study of the entire situation which made it possible to begin practical automatic operation promptly, after the skips were hung in the shaft.

Differences in operating conditions will naturally require different methods of control, so that it is to be expected that for large automatic mine hoists which may be built in the future the control system will differ from the Inspiration system in several respects. However, the exhaustive investigation of the subject which preceded this installation,

and the experience gained in the adjustment and starting of the Inspiration equipment, make it possible to determine readily the feasibility of other proposed automatic hoists for different conditions of operation. The experience thus gained makes it practicable, moreover, to build equipments for greater depths, higher speeds, or other exacting conditions for which, without this experience, it would be impossible to make designs with reasonable certainty of success.

The application of automatic mine hoists will always be limited by the fact that operation cannot be truly automatic, except where the conditions of hoisting are reasonably uniform. In other words, where, under prevailing conditions, the attendance of an operator is required practically continuously throughout the shift in order to change levels, hoist or lower loads out of balance, or handle men, it is impossible to realize any practical advantages by operating automatically during the short periods of hoisting ore regularly from any one level. On the other hand, entire uniformity is not necessary in order to make automatic operation practicable. As an illustration, consider the case of a main hoist serving a few levels, and an auxiliary hoist in the same hoist house handling all men, timbers, supplies, waste, etc., for all the levels served by the main hoist. Conditions of operation may possibly be sufficiently favorable so that if the main hoist is arranged for automatic operation (or for semi-automatic control from the level stations by the skip tender), the operator for the auxiliary hoist will be able to take care of the hand operation required on the main hoist.

It may reasonably be anticipated that from time to time various mine-hoisting projects will come up for consideration in which the possibilities offered by automatic hoisting should by no means be dismissed without investigation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Laboratory Method for Determining the Capacities of Slime-Settling Tanks*

BY H. S. COE, DENVER, COLO., AND G. H. CLEVINGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

ENGINEERS have long recognized the desirability of correlating the data obtained from small-scale slime-settling tests with commercial work as carried on in large tanks. This need, though most apparent in designing new installations, frequently arises also in existent plants, since a large range of experimental work can be performed without interfering with regular operation.

GENERAL SETTLING PHENOMENA

In order to develop rational methods of laboratory testing, it becomes necessary to study the general phenomena of settling. Since it was desired to give the public the benefit of these methods at as early a date as possible, together with the knowledge of the general principles of slime settling necessary for a clear understanding of the laboratory tests, we have ventured to discuss the subject in so far as brought out by the work which we now have under way. We realize that certain of our preliminary conclusions may be subject to modification, but it seems probable that, with but few exceptions, the general laws enunciated will cover the settling behavior of all pulps encountered in metallurgical plants. However, it must be recognized that there is still a vast amount of work to be done upon certain phases of the subject; therefore, at this time we will attempt only a discussion of the behavior of ore pulps during the process of settling or dewatering, under certain given conditions. The laws and principles controlling these conditions are not fully established, so that a complete discussion of them will not be attempted in this paper. In metallurgical practice, slime pulp consists of water, finely divided sand

* This work was begun in the metallurgical laboratory of Stanford University and later transferred to the coöperative metallurgical laboratory of the U. S. Bureau of Mines at the Panama-Pacific Exposition. Recently Mr. Coe has continued the work on behalf of the Dorr Cyanide Machinery Co. This is contribution No. 1 from the hydrometallurgical division of the Exposition laboratory and is published with the permission of the Director of the U. S. Bureau of Mines.

or granular particles, and colloidal material. In this connection water is used as implying either ordinary water or water containing some chemical or chemicals in solution. The chemical or chemicals present which exert an influence upon the subsequent settling behavior of the pulp are known as electrolytes. The electrolyte has the property of causing the colloidal portion of the slime to form aggregates known as flocs, particles having a more or less definite size, consisting of water, colloidal material and usually fine granular material which has been entrapped. The flocs settle in the liquid medium according to certain laws.

THE FOUR SETTLING ZONES

If a thin pulp, of a dilution of, say, 10 to 1, is placed in a 1,000 c.c. cylinder, after thorough mixture, at least momentarily, it forms a homogeneous mass, as shown in Fig. 1(*E*). After a short time, however, it

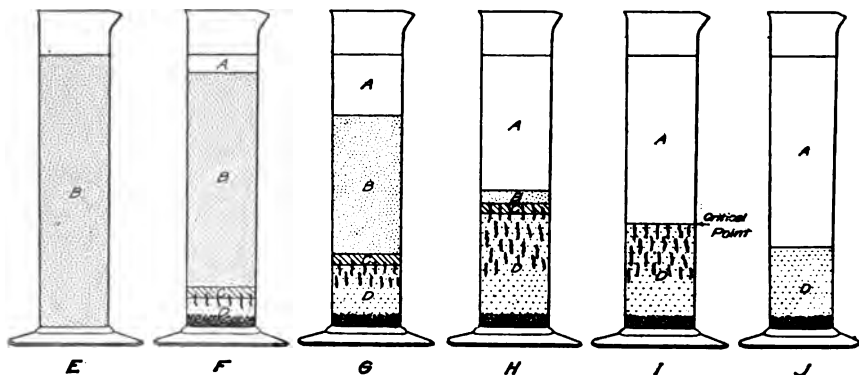


FIG. 1.—EXPERIMENT SHOWING VARIOUS STAGES OF SLIME-SETTLING.

assumes a flocculent structure which, after settling a brief period of time, forms four distinct zones (*A*, *B*, *C*, and *D*), as indicated in Fig. 1 (*F*). The first particles which reach the bottom of the cylinder are the coarser granular sand which may be present in the pulp. Immediately following this and somewhat contemporaneously with the settling of the sand, the slime flocs nearest the bottom settle, filling the interstitial spaces between the sand particles, and build up, one upon another, in a zone of increasing depth. This we term zone *D*, which may be defined as that portion of the pulp wherein the flocs, considered as integral bodies, have settled to a point where they rest directly one upon another. After pulp enters zone *D*, further separation of liquid must come through liquid pressed out of the flocs and out of the interstitial spaces between the flocs. Immediately above zone *D* is a transition zone *C*. The pulp in zone *C* decreases in percentage of solids from the bottom, where the flocs enter zone *D*, to the top, where the consistency of the flocculated pulp is the

same as that of the original pulp. In speaking of flocculated pulp, it is intended to eliminate from consideration the coarser portion of the contained sand which falls directly through the overlying zones into zone *D*. Above *C* is zone *B*, of constant consistency of flocculated pulp and of the same consistency as the flocculated pulp in the feed pulp. Zone *A*, overlying zone *B*, is clear water or solution. In the case of a very rapidly settling slime, particularly with material which has been roasted, zone *A* in the earlier stages may be turbid, due to finely divided matter remaining in suspension. Later this very fine material settles and the liquid becomes clear, although there are cases, especially when the liquid contains very little electrolyte, where it remains turbid for a long time.

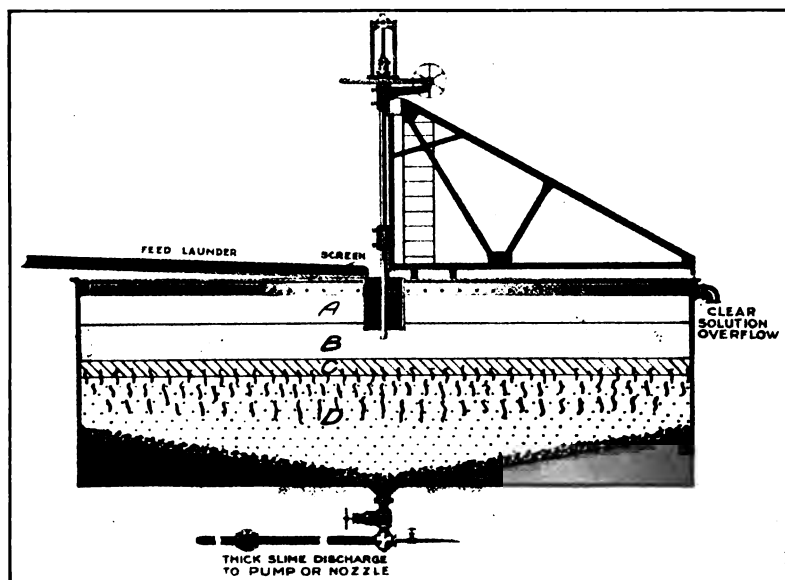


FIG. 2.—DORR THICKENER SHOWING SLIME-SETTLING ZONES.

The effect of the relations between the depths of zones *B* and *C* upon curves derived from settling tests will be discussed later.

Fig. 1 (*E*) shows a cylinder freshly filled with pulp of a consistency of about 10 parts water to 1 part ore. In this illustration zone *B* occupies the total depth. *F*, *G*, and *H* of Fig. 1 show progressive stages of settling in which zones *A* and *D* are growing deeper, zone *B* is decreasing in depth, and zone *C* remains constant—a feature of this particular pulp. Fig. 1 (*I*) shows the condition when all of the pulp has entered zone *D* and compression of the slime flocs is going on. Fig. 1 (*J*) shows the final stage of settling, beyond which the pulp will not thicken further.

With intermittent operation, any one of the stages described may represent the condition in the thickener, depending upon the length of

time that the pulp has been allowed to settle. In the operation of continuous thickeners, the feed of the thin pulp at the center of the tank, the overflow of clear liquid at the periphery of the tank, and the discharge of the thickened pulp at the bottom, are generally continuous. In a continuous thickener, the four zones previously described in discussing intermittent settling are generally present as shown in Fig. 2. At the top there is a zone of clear water, *A*. Beneath this is zone *B*, consisting of flocculated pulp of uniform consistency. Directly beneath this is a transition zone, *C*, and at the bottom a zone, *D*, of pulp which is undergoing compression.

In making tests, the settling rates of thin pulps are determined by readings taken at the juncture of zones *A* and *B*, *i.e.*, where the pulp surface joins the clear liquid.

We designate as free settling pulp all of the pulp in zones *B* and *C*, wherein the sand and flocs are falling freely through the liquid without pressing on the layers of flocs beneath, although it is evident, from the peculiarly interlocking structure of flocculated pulp, that there are points of contact between the flocs even in these zones.

CRITICAL SETTLING POINT DEFINED

We designate as the *Critical Settling Point* the top of zone *D* just as zone *C* disappears. At this point the flocs at the surface just rest upon each other, but compression has not yet commenced in the surface layer. It is therefore obvious that any elimination of liquid from zone *D* cannot be accomplished by free settling but must be effected by compression of flocs. The water liberated by compression finds its way out of zone *D* through tubes or channels which form drainage systems upward through the zone. The trunk channel for any system has its outlet at the top of zone *D*.

Since zone *B*, Fig. 1 (*E*, *F* and *G*) is made up of flocculated pulp of constant consistency,¹ the flocs in this zone will settle at a constant rate so long as the zone exists.

If zone *C* remains shallow and of constant depth, the liquid being expelled from zone *D* may be ejected through zone *C* with little admixture with this zone. This upward current of liquid which has diffused through zone *B* during the first stages of settling may also be ejected through zone *B*, when *B* becomes very shallow, and thereby considerably increase the apparent settling rate of the pulp at this stage. This does not indicate that during this period more pulp passes into zone *D*. It is merely a surface phenomenon, indicating that zone *C* is very shallow.

¹ The constant settling rate in the upper part of the tank has been noted by Allan J. Clark in discussing the settling behavior of the slime at the Homestake. See A Note on the Settling of Slimes, *Engineering and Mining Journal*, vol. xcix, No. 9, p. 412 (1915).

A curve plotted from the results of a settling test such as is illustrated in Fig. 1 will show a constant settling rate throughout the stages represented in Fig. 1 (*E*, *F* and *G*), an increased rate when the stage shown in Fig. 1 (*H*) is reached, followed by a rapid retardation in the settling rate after zone *B* disappears, until zone *C* has also disappeared. Following this is a very slow rate of settling which gradually becomes slower as the water is compressed from zone *D*. The final stage of settling is reached when no further liquid is expelled from zone *D* by compression. The later stages of settling are shown in Fig. 1 (*I* and *J*).

SLIMES CLASSIFIED AS TO SETTLING RATE

The type of settling showing a constant rate down to near the critical point, regardless of whether acceleration is manifested or not, we term

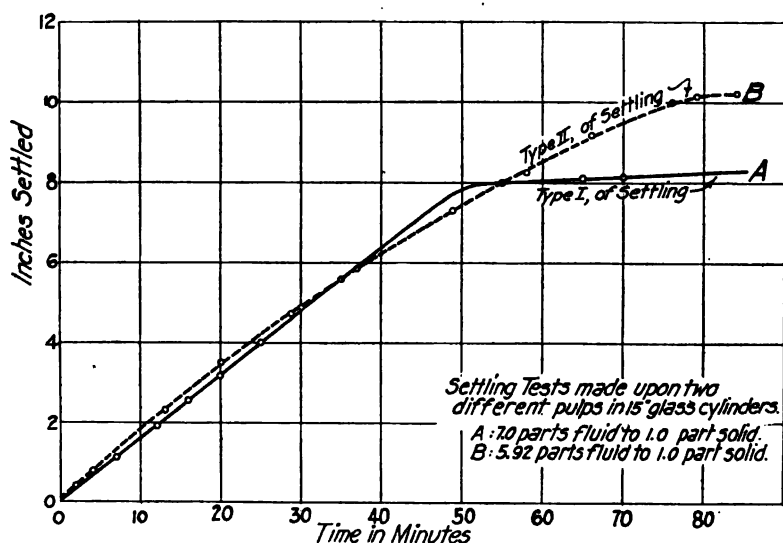


FIG. 3.—CURVES SHOWING SETTLING RATES OF TWO TYPES OF SLIME.

Type I. This is illustrated in Fig. 3 by curve *A*. With certain pulps a constant rate of settling does not occur, since zone *C* rapidly reaches the surface. The surface of such a pulp contained in a cylinder, therefore, settles at a continually decreasing rate down to the critical point. We term this type of settling Type II. It is illustrated by curve *B*, Fig. 3.

Type I of Settling

A pulp containing very little sand was mixed with water containing 40 points of lime (100 points being equivalent to a saturated solution of lime in water, *i.e.*, 0.13 per cent.) to give a pulp of a consistency of 5.7

parts of liquid to 1 part of ore. This pulp settled at a constant rate to near the critical point. Tests were also made to ascertain the settling rates of pulps of 4.71 and 4.00 parts of liquid to 1 part of ore, respectively.

	Consistency of Pulp Parts Water to 1 Part Ore	Rate of Settling Feet per Hour
I.	5.7	0.6
II.	4.71	0.464
III.	4.00	0.414

Conceive a column of pulp consisting of three distinct layers: I (upper), II (middle), and III (lower), having consistencies of 5.7, 4.7 and 4.00 parts fluid to 1 part solids, respectively, layer I feeding into layer II, and layer II feeding into layer III. Let it be assumed that the liquid contained in the flocs and being carried down with them is 3 parts to 1 part of solids. Since the flocs in the three layers are passing downward at known rates, we may compute the weight of solids being fed from a given area of each layer per unit of time, *i.e.*, from layer I to layer II, from layer II to layer III, and leaving layer III, as follows: The pulp in layer I consists of 5.7 parts fluid to 1 part of solids, making a total of 6.7 parts. Of this, 3 parts fluid and 1 part solids are passing downward while 2.7 parts of fluid remain behind. Therefore, for each 2.7 parts of fluid which remain behind, 1 part of solids passes downward and out of layer I. Since the pulp in layer I settled at the rate of 0.6 ft. per hour, the water remaining behind in an area of 1 sq. ft., in 1 hr. would be 0.6 cu. ft., equivalent to 37.41 lb. (water weighs 62.35 lb. per cubic foot) and the weight of solids passing out of layer I per hour would be 13.8 lb. per square foot. In a similar manner it is calculated that the solids passing out of layer II are 16.9 lb. per square foot per hour and the solids passing out of layer III are 25.8 lb. per square foot per hour. The following table shows the weight of solids, in pounds, which may be fed from each layer, assuming the quantity of fluid carried downward in the flocs to be as follows: 1 part fluid to 1 part solids; 2 parts fluid to 1 part solids; $2\frac{1}{2}$ parts fluid to 1 part solids; and 3 parts fluid to 1 part solids.

Layer	1 to 1 Pounds	2 to 1 Pounds	$2\frac{1}{2}$ to 1 Pounds	3 to 1 Pounds
I.	7.95	10.1	11.6	13.8
II.	7.78	10.6	13.1	16.9
III.	8.60	12.9	17.2	25.8

It will be noted that if the flocs are assumed to contain between 2 to 1 and 3 to 1 parts of liquid, the weight of solids passing out of layers II and III will be greater than the weight passing into each from the overlying layer. Therefore, layers II and III must decrease in depth, and finally become infinitely shallow. The rate of settling will therefore

remain constant in a settling test on the pulp of the consistency of 5.7 parts liquid to 1 part solids, until the pulp becomes thicker than that in layer III. If the flocs contained only 1 part liquid to 1 part solids, layer II would increase in depth and in a continuous settling test the rate of settling would decrease until the rate of 0.464 ft. per hour was reached and thereafter would remain constant until all of the pulp became thicker than that in layer III. Since investigation shows that the settling rate remains constant almost to the critical point, the proportion of liquid in the flocs must be greater than 1 to 1.

Type II of Settling

Fig. 3 (B) is the curve plotted from the results of a continuous settling test made upon a pulp showing Type II² settling. This pulp showed a very fine flocculated structure. The consistency was 5.92 parts fluid to 1 part solids. It was placed in a 1,000 c.c. cylinder and allowed to settle without interruption to the critical point. Readings were taken at intervals and the settling rates in feet per hour were computed.

After permitting the sand to settle out of a sample of this pulp, it was found that the free-settling sand represented 46 per cent. of the total solids. Settling rates were determined on pulps of varying consistencies. The consistencies are estimated as the ratio of fluid to total solids (1) and as the ratio of fluid to flocculated solids (2). The consistencies and settling rates are given below:

Layer	Ratio of Fluid to Total Solids (1)	Ratio of Fluid to Flocculated Solids (2)	Rate of Settling in Feet per Hour
I.	5.92-1	10.95-1	1.03
II.	5.17-1	9.57-1	0.89
III.	4.42-1	8.20-1	0.69
IV.	3.67-1	6.79-1	0.52
V.	3.17-1	5.87-1	0.417
VI.	2.92-1	5.40-1	0.372
VII.	2.42-1	4.48-1	0.297
VIII.	2.00-1	3.70-1	0.248

Using the same line of reasoning that was followed in estimating the settling capacities through the various layers in discussing Type I of settling and estimating the amount of fluid contained in the flocs at 2 parts fluid to 1 part flocculated slime and also at 3 parts fluid to 1 part flocculated slime, the following capacities are obtained in pounds per hour of flocculated slime which may pass out of each respective layer:

² The same ore may yield both Type I and Type II of settling under varying conditions.

Layer	Flocs Containing 2 to 1 Pounds	Flocs Containing 3 to 1 Pounds
I.	7.16	8.06
II.	7.32	8.43
III.	6.93	8.26
IV.	6.75	8.55
V.	6.70	9.00
VI.	6.81	9.63
VII.	7.50	12.58
VIII.	9.12	22.40

If we assume that the flocs are made up of 3 parts fluid to 1 part flocculated solids, the estimated weight of solids passing out of each layer appears to be greater than the amount fed into it from above. In this event the lower layers could not exist and the settling would, therefore, follow Type I. If we assume flocs made up of 2 parts fluid to 1 part solids, the estimated weight of solids passing out of each layer (eliminating the settling rate in II, which was probably erroneous) into the layer below becomes less than the amount fed into it down to layer VI, after which the quantity of solids passing out of each layer is greater than the weight fed into it from above. In this case layers V, IV, III, etc., would increase in depth as layers I, II, etc., disappear, and the settling rate indicated by a continuous settling test made in a cylinder, filled with pulp similar to that in layer I, will decrease continuously until the pulp at the surface is of a consistency between that of the pulp in layers V and VI. Layers VI to VIII will be infinitely thin, since it is possible for more pulp to pass out from each layer than the amount which enters it and, therefore, the settling rate of pulps of the consistencies contained in these layers will only appear in a period of rapid transition from the settling rate in layer V to the settling rate in some layer below VIII. It has been calculated, from the data from which curve *B* (Fig. 3) was plotted, that the rate of settling decreased in 66 min. from 1.03 ft. per hour, as in layer I to 0.55 ft. per hour, and in the next 18 min. dropped to 0.08 ft. per hour. From the above it should be evident that the weight of solids which can pass from each layer is dependent upon the percentage of sand present (which passes downward carrying no liquid), upon the ratio of fluid to solids in the flocs and upon the settling rate of the pulp in the layer considered. The capacity of a layer of any consistency of flocculated pulp to discharge its flocculated solids may be calculated by the following formula:

$$c = \frac{62.35r}{f - d}$$

c = capacity in pounds of solids per square foot per hour.

r = settling rate in feet per hour in the layer.

f = ratio of fluid to flocculated solids in the layer.

d = ratio of fluid to flocculated solids in the flocs.

62.35 = weight of 1 cu. ft. of water.

In the illustration of Type II of settling, pulp of a consistency of 3.17 parts fluid to 1 part solids or 5.87 parts fluid to 1 part flocculated solids settled at a rate of 0.417 ft. per hour. Assuming that the flocs contain

2 parts fluid to 1 part solids, the above formula gives $\frac{62.35 \times 0.417}{5.87 - 2.0} = 6.7$

lb. per square foot per hour of flocculated solids and 13.4 lb. of fluid passing downward out of layer V, but since the flocculated solids represent only 54 per cent. of the total solids, the total capacity would be 12.4 lb. per square foot per hour. The proportion of fluid carried down in the flocs to total solids would be 13.4 : 12.4 :: 1.08 : 1. Given the ratio of fluid to total solids in a pulp with a known settling rate and the ratio of fluid to total solids to be discharged, it is not necessary to know the percentage of fluid in the flocs in order to determine the maximum capacity of such a layer of pulp to discharge pulp of the consistency required. Thus in the above case the ratio of fluid to total solids equals 3.17 to 1, and the ratio of fluid to solids in the discharge required is 1.08 to 1. Ap-

plying the formula, we find $c = \frac{62.35 \times 0.417}{3.17 - 1.08} = 12.4$ lb., which is the same as that given above. The formula may, therefore, be written

$$C = \frac{62.35R}{F - D}, \text{ where}$$

F = ratio of fluid to solids in the pulp tested.

R = rate of settling in feet per hour of a free-settling pulp of consistency F .

D = ratio of fluid to solids in the discharge required.

C = capacity in pounds of solids per square foot per hour, which may be discharged with a consistency of D from a layer of pulp of a consistency of F settling at a rate R .

APPLICATION OF THE FORMULA IN DETERMINING THE CAPACITY OF CONTINUOUS THICKENERS

Since in any vessel used for settling there are layers of pulp of every consistency between that of the feed and that of the discharge, although some of the layers may be infinitely shallow, the maximum capacity in pounds of solids per square foot per hour is expressed by the formula

$C = \frac{62.35R}{F - D}$, and the maximum capacity possible will be the smallest value of C obtained by applying this formula to a series of tests made upon pulp ranging in consistency from that of the feed to that of the

thickest free-settling pulp, and taking for D the ratio of fluid to solids desired in the discharge.

THICKENING IN ZONE D AND CONSISTENCY OF DISCHARGE POSSIBLE

We have established through numerous experiments the fact that after pulp reaches the consistency where the flocs touch each other, further elimination of water becomes approximately a function of time, in so far as tests for metallurgical practice are concerned. It seems probable that the relationship of pulp consistency to time of thickening is an indirect one, depending upon the effect of compression caused by the depth of the pulp being counteracted by the resistance in the pulp to the escaping water, and to the admixture of the pulp in the upper portion of the thickening layer with the ascending water from the lower layers. A large number of comparative tests have been made in vessels of from 1 to 10 ft. in depth. If portions of pulp be placed in vessels of various depths, it will be found that the critical point will be reached earliest by the surface of the pulp in the most shallow vessel, since the pulp does not have so far to settle to reach the zone of compression (D), but the settling rates in layers of like consistency are the same for vessels of all depths. In the deep vessels the critical point will be proportionately lower than in the shallow vessels. This may be explained by the fact that the average time of compression in the thickening zone before the critical point is reached is greatest for the pulp in the deep vessel. If the fluid be expelled as a function of time after the flocs enter the zone of compression, it is but natural that this should be the case.

In two vessels of unequal depth, filled with the same kind of pulp and started to settle at the same time, the total pulp in the more shallow vessel will begin to compress more quickly and will thereafter for many hours remain thicker than the pulp in the deeper vessel, even though the critical points occur not more than an hour apart. This indicates that thickening in the compression zone is a function of time. In one or two cases it was noted that after a time pulp in the deeper vessel became the thickest, but, upon decanting the fluid from the pulp contained in the shallow vessel and stirring the remaining pulp, settling was resumed until the pulp was as thick or thicker than that in the deeper vessel. We assume that this difference was due to the fact that most of the sand in the shallow vessel reached the bottom, while in the deeper vessel some of the sand lodged in the lower part of the compression zone and assisted compression.

Another series of tests was made by placing moderately thick pulp in cylinders of various depths. It was found that after allowing time for the structure to develop and channels to form in the deep cylinder, the total settling at various periods of time was approximately propor-

tional to the depth of the cylinder. There is, however, a limit to this, for it appears that as the settling by compression in a deep vessel reaches the slowest settling rate for free-settling pulp, a zone of semi-thickened pulp will be formed and maintained until the compression rate for the total column slows down to a lower rate than the lowest free-settling rate of the pulp. In this series of tests the greatest consistency for various periods of time is almost invariably reached in the most shallow cylinders, but the difference is slight and to a great extent due to the resistance of the pulp to liquid rising from the bottom of the cylinder. In practice in continuous thickening, such a zone of semi-thickened pulp never reaches the bottom of the tank, since large amounts of fluid do not have to be liberated from the pulp at the bottom. This resistance is therefore of less consequence in practical operation.

The diameter of the cylinder or tank used for making the test seems to have little or no influence upon settling. Tests made upon various pulp samples in cylinders varying in diameter from $\frac{1}{2}$ in. to $2\frac{1}{2}$ in. show very slight differences in the settling rates and in the final consistency of the settled pulp. The $\frac{1}{2}$ -in. tube generally gave a more rapid rate of settling down to the critical point, probably due to convection currents. Beyond this point, the pulps in the tube of small diameter settles more slowly. The following experiment will illustrate this point: A 4-ft. cylinder, $2\frac{1}{2}$ in. in diameter, and a 4-ft. cylinder, $\frac{1}{2}$ in. in diameter, were filled with Liberty Bell pulp, 3.26 parts water to 1 part dry slime. The water contained 4 points of lime. The consistency of the settled pulp, after settling 48 hr. in the $2\frac{1}{2}$ -in. cylinder was 1.8 parts water to 1 part of dry slime, while in the $\frac{1}{2}$ -in. cylinder it was 1.825 parts water to 1 part dry slime. It was found that the difference was greater when the water contained a greater proportion of electrolyte (in this case, lime), for the reason that as the proportion of electrolyte is increased, the slime flocs become larger and agglomerate in such a manner as to offer more resistance to settling in the small cylinder. With a large proportion of lime, the pulp in the small cylinder settled in divisions, due to the arching effect of the large slime flocs, there being alternate layers of clear liquid and of thick pulp. It was noted that ultimately the sum of the zones of clear liquid equaled the total depth of clear liquid formed in the larger cylinder. All of our tests have indicated that if the diameter of the cylinder is 2 in. or over, little or no influence is exerted upon the rate of settling.

Table I gives the record of some of the tests made to establish the fact that ultimate consistency of the pulp is a function of the time during which the pulp remains in the thickening zone. The names of the mines from which these pulps came are not given, as permission to publish them was not obtained. Example *G* in Table I was made on a pulp consisting of 6 parts solution of zinc sulphate containing hydrogen sul-

phide to 1 part precipitated barium sulphate and zinc sulphide. Despite the low percentage of solids, the pulp is very viscous and is not free settling. Air bubbles remained entrapped for 36 hr. when the pulp was allowed to settle in a cylinder 16 in. in depth. In a cylinder 56 in. in depth the bubbles were ejected and the settling rate became 50 per cent. greater than the ratio of the depth to that of the lower cylinder would indicate. It would seem that in this type of pulp the law of compression being a function of time would not hold. Apparently the reason is that the highly viscous homogeneous structure resists the formation of tubes and channels in zone *D*, except under higher pressure through a deeper column of pulp. It is possible that with a large proportion of electrolyte, for instance lime, certain ore pulps also may behave in this manner. When such cases arise, recourse must be had to the test described later in the paper. Such a condition is generally indicated by the absence of the ordinary flocculated structure and the fact that active channeling does not take place in zone *D*.

TABLE I.—*Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights*

Example A

Type of Pulp Siliceous; 60 Per Cent. through a 200-mesh Screen

Time of Settling, Hours	Height of Cyl. 91.25 In. Settlement in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 44.75 In. Settlement in Inches	Consistency of Pulp. Fluid to Solid	Height of Cyl. 14.3 In. Settlement in Inches	Consistency of Pulp Fluid to Solid
0	0.00	2.95-1	0.00	2.95-1	0.00	2.95-1
1	4.25		3.92		3.60	
2	9.00		9.22*		4.76	
4	19.20*		14.70		5.90*	1.58-1
19	37.90	1.56-1	19.50	1.50-1	7.80	1.14-1
24	39.90	1.50-1	20.40	1.43-1		
28	41.80	1.42-1	20.40†	1.43-1		
44	43.75	1.38-1	22.60	1.27-1	7.80	1.14-1
	* Tubes to the surface.		* Tubes to the surface. † Drew off water and stirred.		* Drew off water and stirred.	

Example B

Time of Settling, Hours	Height of Cyl. 29.75 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 3 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0	0.00	3.00-1	0.00	3.00-1
16			1.56	1.18-1
24	17.37	1.05-1	1.78	1.00-1
32	18.50	0.90-1	1.83	0.95-1

TABLE I.—*Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights—(Continued)**Example C*
Siliceous Ore

Time of Settling, Hours	Height of Cyl. 123 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 14.3 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0	0	4.73-1	0.0	4.73-1
18	113	1.05-1	13.15 Removed H ₂ O and stirred.	1.08-1
36			13.37	0.90-1

Example D
Very Colloidal Siliceous Pulp

Time of Settling, Hours	Height of Cyl. 92 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 41.5 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 14.1 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0	0.0	2.13-1	0.0	2.13 -1	0.00	2.13-1
17	30.3	1.30-1	14.8	1.235-1	5.57	1.13-1
20	31.0	1.28-1	15.1	1.219-1	5.72	1.11-1
64	36.3	1.13-1	18.4	1.020-1	6.64	0.95-1

Example E
Liberty Bell Pulp Oxidized with a High Percentage of Clay; CaO Traces

Time of Settling, Hours	Height of Cyl. 114 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 45 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 11 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0		3.26-1		3.26-1		3.26-1
1	1.5		1.5		1.56	2.75-1
5	9.4	2.95-1	9.75	2.47-1	3.92	1.97-1
23	45.5	1.81-1	19.5	1.69-1	5.25	1.53-1
29	51.5	1.62-1	21.0	1.56-1	5.39	1.49-1

Example F
Enterprise Ore—Siliceous

Time of Settling, Hours	Height of Cyl. 60 In. Settling in Inches	Consistency of Pulp Fluid to Solid	Height of Cyl. 10 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0.00	0.00	7.00-1	0.00	7.00-1
0.50	4.63		4.50	3.68-1
3.00	24.43		6.18	2.43-1
4.75	35.78	2.61-1	6.50	2.20-1
18.00	41.27	1.93-1	7.10	1.76-1

TABLE I.—Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights—(Continued)

Example G

A Chemical Pulp, Showing an Apparent Exception

Time of Settling, Hours	Height of Cyl. 56 In. Settling in Inches	Height of Cyl. 15.75 In. Settling in Inches	Consistency of Pulp Fluid to Solid
0	0.0	0.0	6-1
7	5.9	1.2	
24	11.3	2.5	

SUMMARY OF SLIME-SETTLING RESULTS

The results of our investigation of slime settling may be summarized as follows:

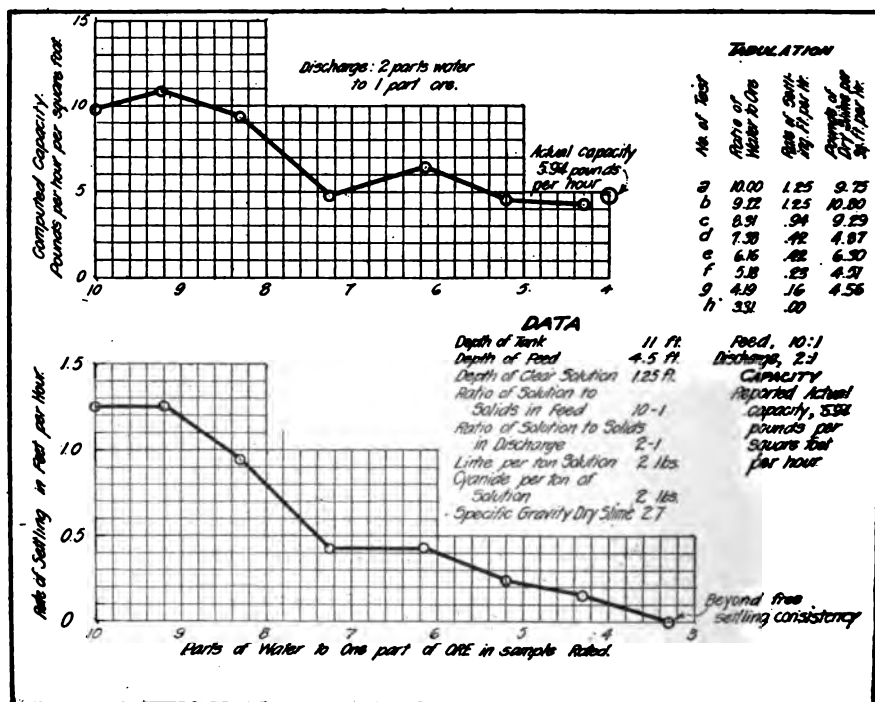


FIG. 4.—SLIME-SETTLING DATA, LIBERTY BELL MILL, TELLURIDE, COL.

1. In thickening pulps which are to be discharged at a consistency such that the discharge is still in the form of free-settling pulp, the depth of the tank used is of no consequence, except in so far as it permits a depth of feed sufficient to avoid surface agitation and allows ample depth of clear liquid to care for fluctuations of the feed and changes in

the character of the pulp in the case of continuous thickeners, and sufficient depth to give ample capacity to avoid the necessity of frequent charging and discharging in the case of intermittent thickeners.

2. When thickening pulps to a consistency where it is necessary to expel fluid by compression, sufficient capacity must be given the tank so that the pulp will be retained the necessary period of time to thicken it to the required density and also to allow sufficient storage to compensate for fluctuations in the feed and discharge. This capacity may be obtained by varying either the depth or diameter to give the required cubical content.

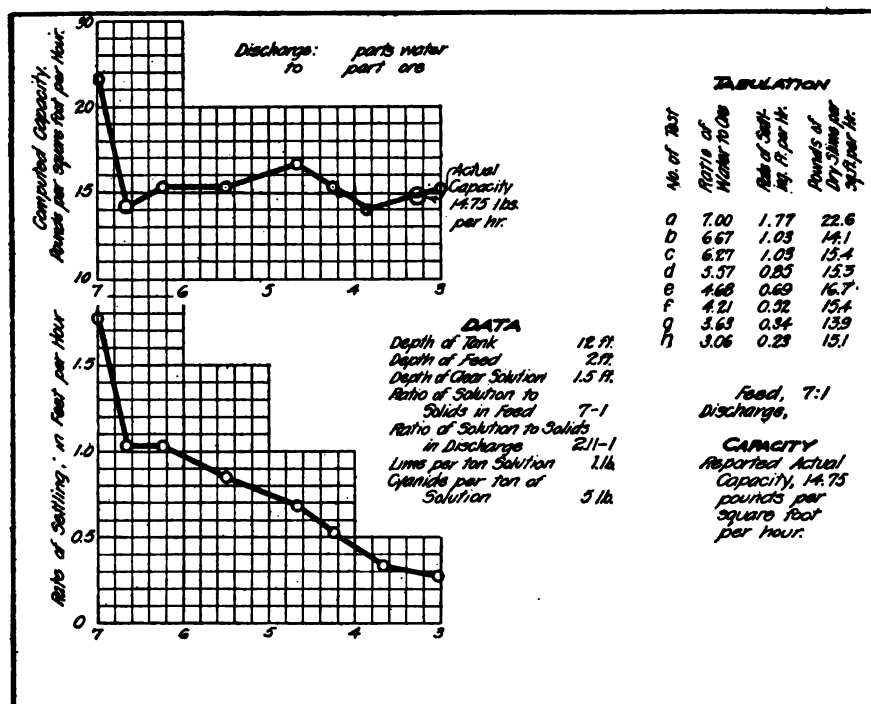


FIG. 5.—SLIME-SETTLING DATA, BELMONT MILL, TONOPAH, NEV.

3. The consistency of discharge possible may be closely determined by allowing a cylinder of thick but free-settling pulp to settle, taking readings at intervals of a few hours up to the point where settling practically ceases.

4. The required area may be computed by applying the formula

$$A = \frac{2,000}{24 \left(\frac{62.35R}{F - D} \right)}$$

A is the area in square feet required to thicken 1 ton of 2,000 lb. of solids to a consistency in the discharge of D (parts fluid

to 1 part solids by weight), per day of 24 hr. A series of settling rates is taken on pulps ranging in consistencies from that of the proposed feed to the thickest free-settling pulp. D is taken as the ratio of fluid to solids in the thickest pulp which can be economically obtained. The greatest value for A indicated by the tests is taken as the required area. Under ordinary circumstances a factor of safety should be allowed over the calculated area to take care of changes in the character of the pulp and variations in temperature. It will be noted that this is the same formula previously

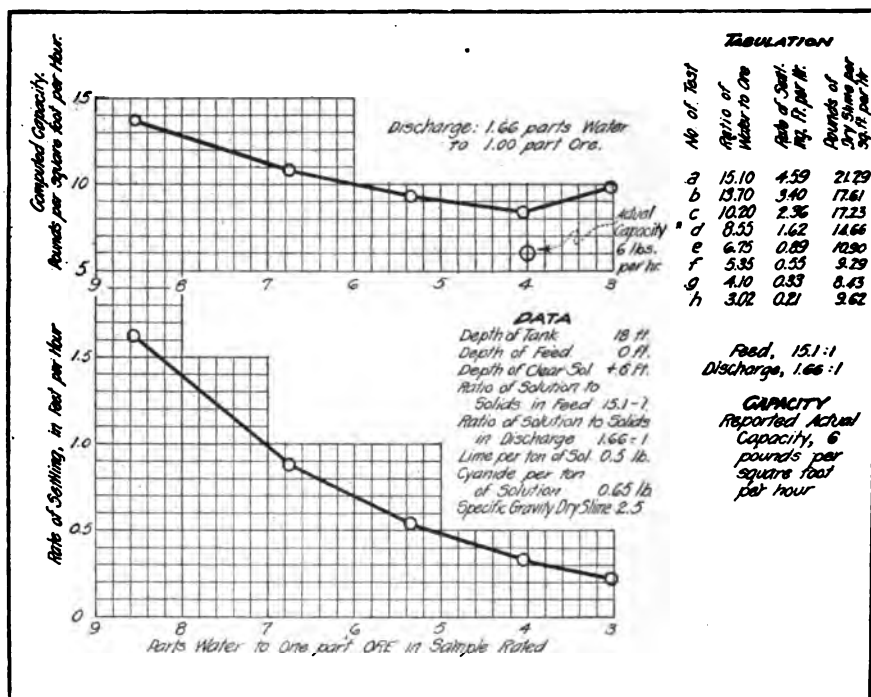


FIG. 6.—SLIME-SETTLING DATA, PORTLAND MILL, VICTOR, COL.

given, modified to give area required instead of capacity per square foot per hour.

5. The required depth of the thickener may be ascertained by computing the capacity of the thickening zone to contain a supply of solids equal to the total capacity of the tank for the number of hours required to thicken the pulp to the density required in the discharge, and to this depth adding an allowance for the lost space due to the pitch of the drag in the thickener; also from $1\frac{1}{2}$ to $2\frac{1}{2}$ ft. for depth of feed and a further allowance for storage capacity when the discharge may be closed. The following is an illustration of the method used in computing the area and depth required in a thickener to handle 100 tons of pulp per day, thicken-

ing from 6 parts fluid to 1 part solids down to 1.12 parts fluid to 1 part solids.

Test	Parts Fluid to 1 Part Solids	Rate of Settling in Feet per Hour	Area Required per Ton of Solids per 24 Hours
1	6.00	2.180	3.00
2	4.94	1.190	4.29
3	4.00	0.893	4.31
4	3.51	0.758	4.22
5	3.00	0.600	4.05

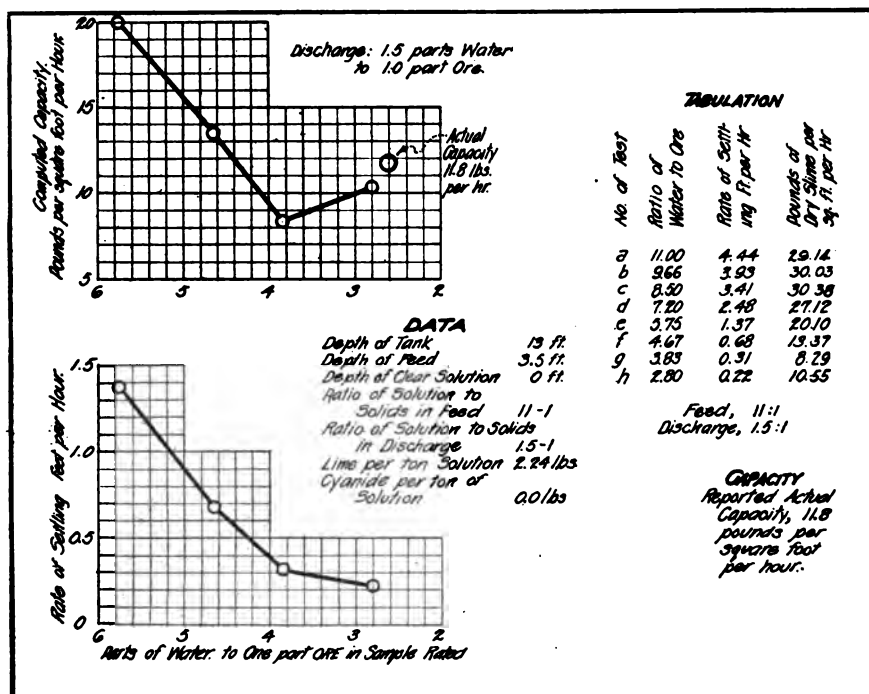


FIG. 7.—SLIME-SETTLING DATA, NIPISSING LOW-GRADE MILL, COBALT, ONT.

The formula $A = \frac{2,000}{24 \left(\frac{62.35R}{F-D} \right)}$ was applied to ascertain the above

areas. Taking for instance the third test, $R = 0.893$, $F = 4$, $D = 1.12$, then $A = \frac{2,000}{24 \times 62.35 \times 0.893} = 4.31$, the number of square feet required

per ton per day. For handling 100 tons per day 431 sq. ft. or the area of a tank 23.4 ft. in diameter would be required. In the above example the third test is taken for the required area as being the largest area indi-

cated by any of the tests, while the areas indicated by the fourth and fifth tests show that the areas are diminishing for the thicker pulps, therefore the turning point has been passed.

To Ascertain the Necessary Depth of Tank

Assuming that the following results are obtained from settling tests made in a cylinder 12 in. deep upon 3 to 1 pulp, the computations are made as follows:

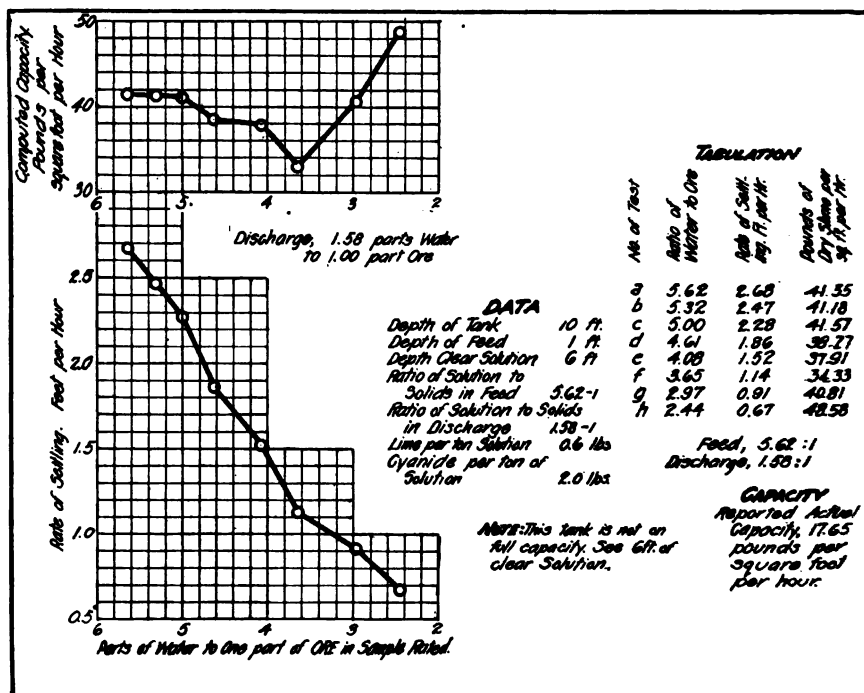


FIG. 8.—SLIME-SETTLING DATA, PRESIDIO MILL, SHAFTER, TEX.

Time of Thickening, Hours	Consistency Fluid to Solids
2	1.70:1
4	1.59:1
9	1.35:1
14	1.20:1
19	1.12:1

Since an area of 4.31 sq. ft. is required per ton of solids per 24 hr., the total solids per square foot retained in the thickening zone must be $\frac{19 \times 2,000}{24 \times 4.31} = 367$ lb., or 19 hours' supply per square foot. There is required a 5-hr. supply, of each of the pulp consistencies 1.16 to 1; 1.275

to 1; 1.47 to 1, and a 4-hr. supply of a 1.7 to 1 pulp. The solids per cubic foot in the above pulps are 43.2 lb., 37.6 lb., 33.7 lb. and 30 lb., respectively. The depth of each class of pulp would therefore be 2.23 ft., 2.57 ft., 2.87 ft., and 2.58 ft., or a total depth of 10.25 ft. To this depth must be added a foot for the loss due to the pitch of the drags in the thickener and 1.5 ft. for the depth of the feed, since the feed is thick and the volume will be proportionately low. The total calculated depth of the tank required would be 12.75 ft. If proper allowance were made for storage capacity, the tank might be inconveniently deep. In this case it would be better practice to make the tank 12 ft. deep and make

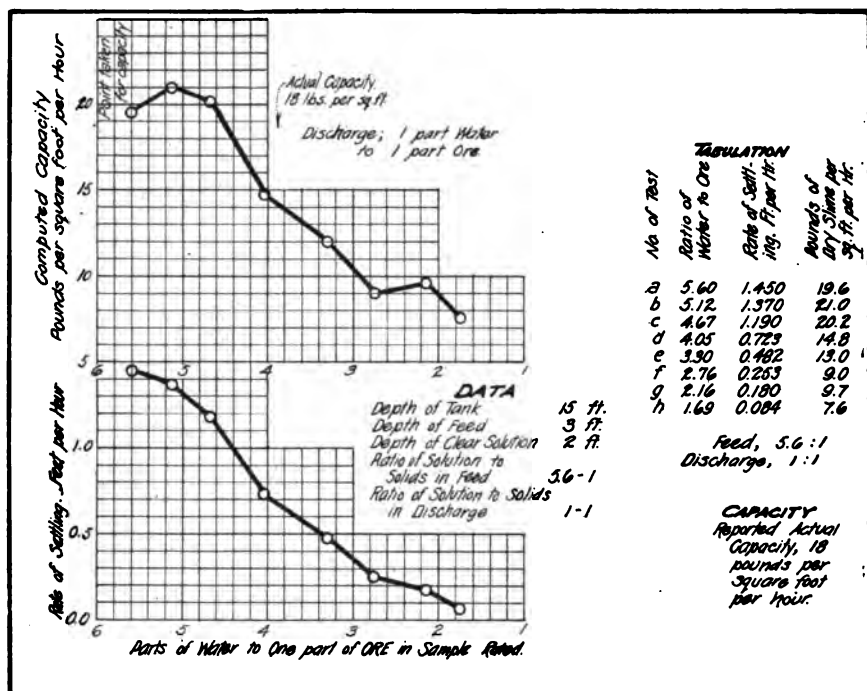


FIG. 8.—SLIME-SETTLING DATA, HOLLINGER MILL, TIMMIN, ONT.

various allowances for additional capacity by increasing the diameter of the tank to give a 30 per cent. increase in area. Ordinarily in settling ore pulps composed of fine granular material with a considerable proportion of colloidal material which may vary in character, it is unwise to provide less than 6 sq. ft. of settling area per ton of solids to be settled daily. Frequently much larger areas are required.

If, in the tests previously cited, the fourth and fifth settling rates had decreased so as to cause the indicated required areas to increase gradually, the solution of the problem would be less simple, for it is necessary to be certain that the settling rate for the thickest free-settling

pulp has been determined and also that the pulp being tested is free settling. Ordinarily the latter point is indicated by the evenly flocculated appearance of the pulp surface, without channels of fluid coming through, and by the uniform texture of the pulp as seen in a glass cylinder. The settling rate, after flocculation is complete, should be constant or gradually diminishing to the critical point. In certain thick, free-settling pulps containing a large proportion of electrolyte, the whole mass agglomerates upon shaking, and in a shallow cylinder the time required to form the natural structure will be so great that the pulp will have passed the critical point before a reliable settling rate has been observed.

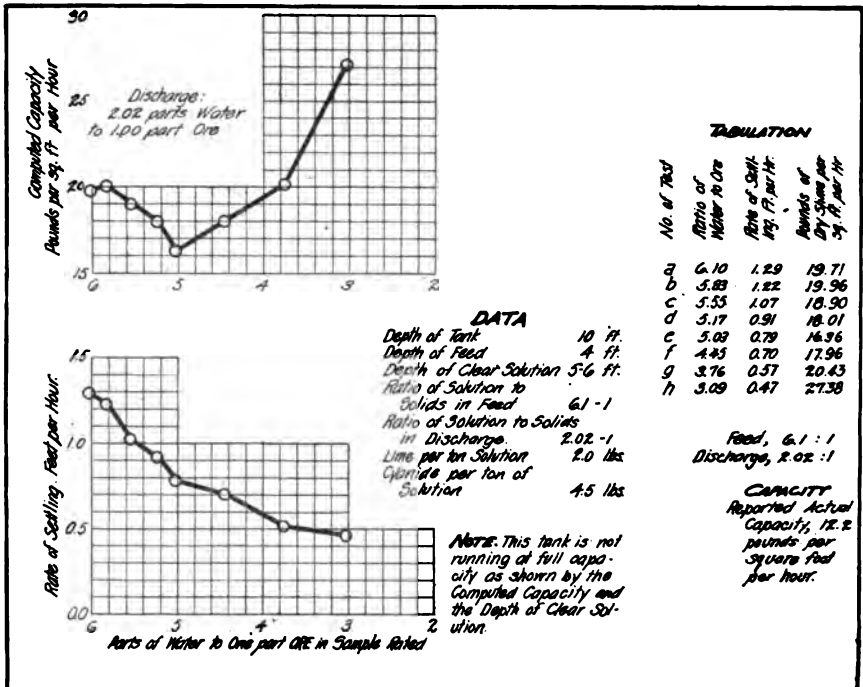


FIG. 10.—SLIME-SETTLING DATA, WEST END CONSOLIDATED MILL, TONOPAH, NEV.

In such cases the series of tests should be repeated using less electrolyte, the object being to ascertain the consistency at which the indicated area establishes itself as a constant, or decreases. Should it not be feasible to reduce the proportion of electrolyte, cylinders 3 or 4 ft. in height may be employed in repeating the tests, in order that sufficient time may be given for the flocculated structure and even texture to become apparent. If, after applying these tests, it is desired to verify the results still further, it becomes necessary to make a test representing actual continuous feed and discharge conditions, as hereafter described in this paper.

In making settling tests the sample should be carefully taken in order that it may be truly representative of the pulp to be settled. The ratio of fluid to solids may be determined by drying a sample from each test. The pulp is placed in a 1,000 c.c. cylinder and well shaken; it is then allowed to settle. Readings should be taken at intervals of from 1 to 3 min. until the pulp has reached a uniform or decreasing settling rate, the maximum rate being taken as the one sought. The readings are most conveniently taken in cubic centimeters and the settling rate in feet per hour may be computed from those by establishing the value

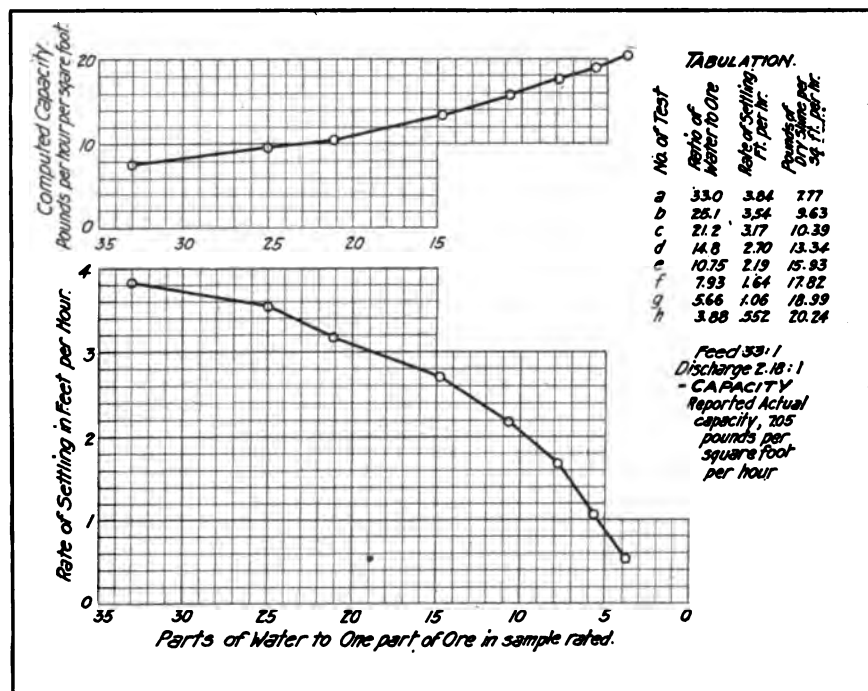


FIG. 11.—SLIME-SETTLING DATA, HOMESTAKE MILL, LEAD, S. D.

in decimal parts of a foot of each cubic centimeter. This is done by measuring, in feet, the length of the 1,000 c.c. graduation on the cylinder and dividing by 1,000.

METHOD OF MAKING SETTLING TESTS

The method of making the settling tests hereafter quoted was as follows: In each case the pulp was permitted to settle $\frac{1}{8}$ in. before the readings were taken.

(a) 1,000 c.c. feed pulp settled for a 2-min. interval.

(b) After completing test (a) and thoroughly mixing by shaking,

25 c.c. of pulp was removed from the cylinder and replaced by 25 c.c. of discharge pulp. After thoroughly mixing, allowed to settle for an interval of 2 min.

(c) 30 c.c. of pulp removed from (b) and replaced by 30 c.c. of discharge pulp, settling interval 2 min.

(d) 45 c.c. of pulp removed from (c) and replaced by 45 c.c. of discharge pulp, settling interval 3 min.

(e) 75 c.c. of pulp removed from (d) and replaced by 75 c.c. of discharge pulp, settling interval 3 min.

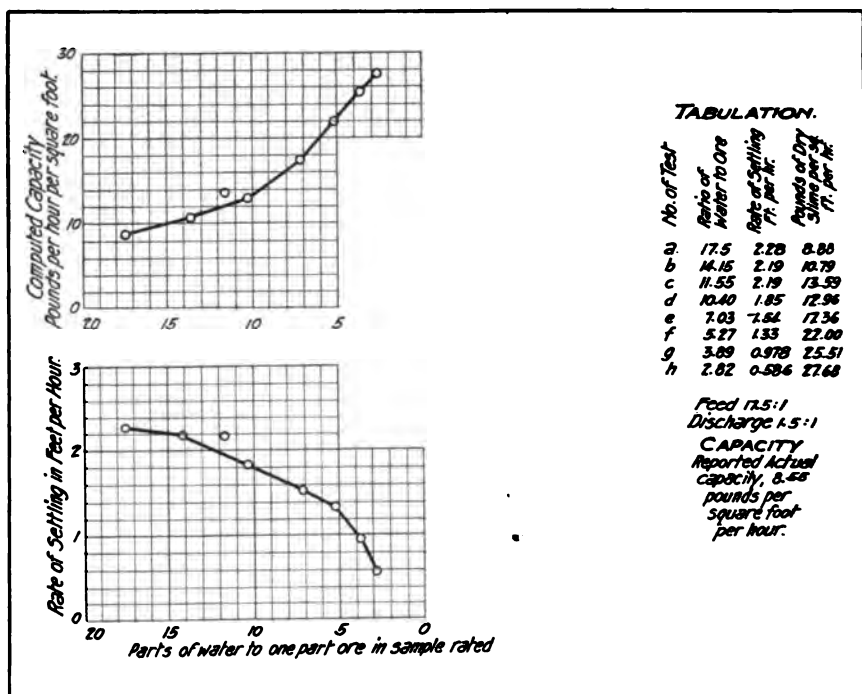


FIG. 12.—SLIME-SETTLING DATA, HOMESTAKE MILL, LEAD, S. D.

(f) 100 c.c. of pulp removed from (e) and replaced by 100 c.c. of discharge pulp, settling interval 4 min.

(g) 160 c.c. of pulp removed from (f) and replaced by 160 c.c. of discharge pulp, settling interval 4 min.

(h) 260 c.c. of pulp removed from (g) and replaced by 260 c.c. of discharge pulp, settling interval 6 min.

Throughout a series of tests the precaution must be taken of thoroughly mixing the contents of the cylinder both before removing pulp and before starting the test on the pulp of altered consistency.

The curves in Fig. 11, showing the settling behavior of Liberty Bell pulp, illustrate the method of plotting used in presenting the results of the tests which have been taken as a basis of comparison with the performance of commercial thickeners. In the lower curve the settling rate in feet per hour is taken as the ordinate, while the ratio of water to dry slime is taken as the abscissa. Each point shown on the curve represents an individual test made upon slime of the dilution noted. The tests appear upon the curve from left to right in the order performed, which places the highest dilution at the origin.

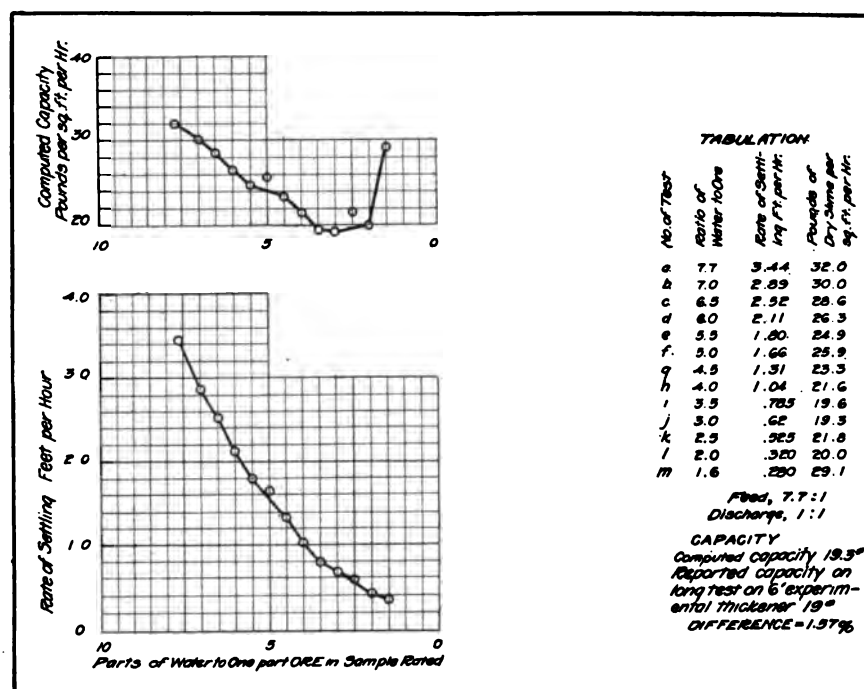


FIG. 13.—SLIME-SETTLING DATA, GOLDEN CYCLE MILL, COLORADO SPRINGS, COL.

In the upper curve, the computed capacity in pounds per hour of dry slime per square foot of settling area is taken as the ordinate, while the dilution is taken as the abscissa, as in the lower curve.

The following table gives the actual capacities, together with the computed capacities, of thickeners used at the various mills enumerated, as shown in curves, Figs. 4 to 13, inclusive. In each case the tests were made at the mill by the local staff, according to directions supplied by us. From the data received the capacities were computed, employing the method described in the paper.

Curve	Pulp	Computed Capacity, Pounds per Square Foot	Actual Reported Capacity, Pounds per Square Foot	Number of Feet of Clear Solution Reported	Consistency of Feed Pulp	Consistency of Discharge Pulp
Fig. 5	Liberty Bell	4.87	5.94	1.25	10.00-1	2.00-1
6	Belmont...	14.1	14.75	1.5	7.00-1	2.11-1
7	Portland (low-grade)*	8.3	6.0	6.0	15.10-1	1.66-1
8	Nipissing (low-grade)	8.24	11.8	0.0	11.00-1	1.50-1
9	Presidio†...	33.0	17.65	6.0	5.62-1	1.58-1
10	Hollinger....	19.7	18.0	2.0	5.60-1	1.00-1
11	West End†.	15.24	11.97	5 to 6	6.10-1	2.02-1
12	Homestake.	7.78	7.05		33.00-1	2.18-1
13	Homestake.	8.86	8.55		17.50-1	1.50-1
14	Golden cycle (roasted ore)	19.3	19.1	0.0	7.70-1	1.00-1

Special Apparatus for Small-Scale Continuous Settling Tests

Although, as demonstrated, reasonably accurate results may be obtained by settling tests made in small glass cylinders (provided the results are properly interpreted), we recommend for testing a special device in which tests may be carried out under the same conditions of depth as would obtain in practice. Since the diameter of the cylinder, if over 2 in. does not in any way affect the settling rate, it is possible to construct from standard iron pipe and fittings, available in most localities, a device which will give results exactly parallel to those to be expected in practice. Fig. 14 shows in detail such a device, which in our experimental work has given very satisfactory results. The body or container is a piece of 4-in. black, iron gas-pipe, threaded at both ends (galvanized iron pipe or fittings should be avoided) on one end of which is screwed a 4-in. pipe flange, while on the other end is screwed a special cast-iron cone. Into the bottom of this cone is screwed a $\frac{1}{2}$ -in. nipple and connected to this is a $\frac{1}{2}$ -in. iron plug cock, into which is screwed another $\frac{1}{2}$ -in. nipple to prevent spattering. If it is not possible to procure the cone-shaped casting for the bottom, the same effect may be obtained by plugging the bottom of the pipe and screwing the discharge valve into a hole tapped at the center of the plug. A thin sheet-iron cone is

* *Portland*.—The bulk of this slime settles readily but there is a small amount of colloidal material which has a tendency to remain in suspension after the bulk of the pulp has settled. In order to obtain as clear solution as possible for precipitation, the thickener was operated below capacity, as indicated by the fact that there was 6 ft. of clear solution.

† *Presidio and West End*.—These thickeners were being operated very much below capacity, as indicated by the fact that there was 5 to 6 ft. of clear solution.

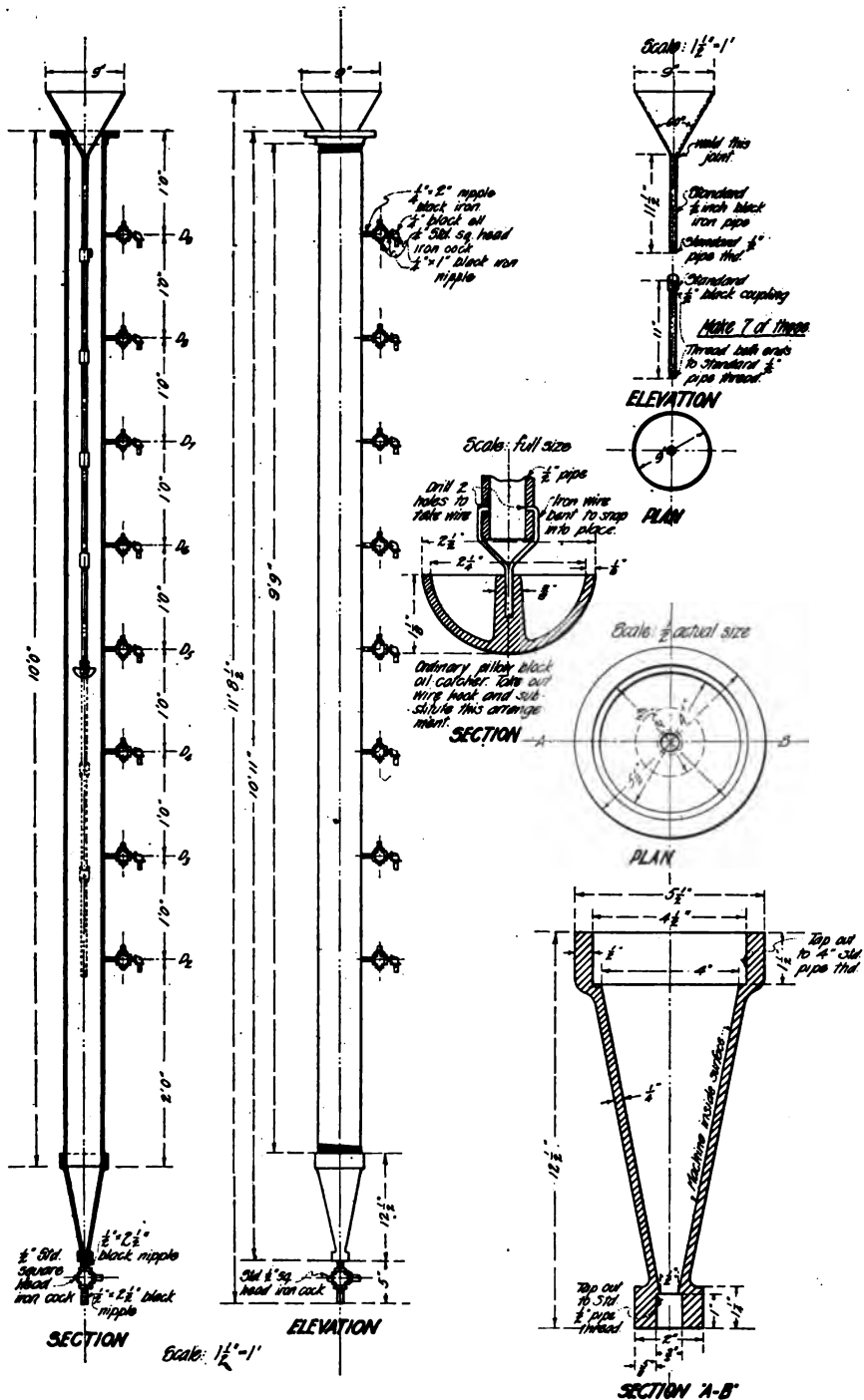


FIG. 14.—APPARATUS FOR SMALL-SCALE CONTINUOUS SETTLING TESTS.

made of such a size that the top just fits the pipe, while the apex is of the same size as the hole in the plug. This cone is slipped inside of the pipe and downward until it rests upon the bottom. Along the side of the 4-in. pipe, at regular intervals of 12 in., are bored holes tapped for a $\frac{1}{2}$ -in. thread. Into each of these is screwed a $\frac{1}{2}$ -in. nipple, to which is attached a $\frac{1}{2}$ -in. plug cock; to this, in each case, is attached an elbow and nipple to divert the outflow downward. The 4-in. pipe is arranged with a tripod or other suitable means of support so that it is maintained in a vertical position at a sufficient distance from the floor for the thickened pulp to be withdrawn readily from the plug cock at the bottom. Resting on top of the main pipe or container is a funnel for introducing the pulp. Attached to this funnel by means of iron sleeves are 11-in. lengths of black iron pipe, the idea being to make each section, with the sleeve when screwed into position, 12 in. in length and so arranged that the outlet from the stem of the funnel is somewhat below the particular cock on the side of the apparatus in use. Any depth of feed in multiples of 12 in. may be obtained by screwing on to the funnel tube the necessary number of 12-in. units. To the lowermost unit in use is attached a cup-shaped iron casting for the purpose of distributing the flow of pulp. This may be an ordinary $2\frac{1}{2}$ -in. cast-iron drip cup, such as is frequently used beneath bearings, attached in such a manner as not to obstruct the outlet of the pipe. The apparatus shown in Fig. 14 has a maximum available depth of 9 ft. for the thickening zone. Therefore, settling tests may be performed in steps of 1 ft., representing what may be expected in tanks where the thickening zone varies from 2 to 9 ft. in depth. If a study of the behavior of deeper tanks is desired, a longer containing pipe may be employed. To illustrate the use of this piece of apparatus: Assume that the settling behavior of a given pulp and the capacity per square foot of settling area of a tank having a thickening zone 5 ft. in depth are to be determined. First make a preliminary settling test in a glass cylinder according to the directions previously given. Let it be assumed that the results are as follows: The critical point in the glass cylinder is ascertained to occur at a dilution of 3 parts fluid to 1 part solid and the mean settling rate down to this point is 1 ft. per hour. After these data are obtained, proceed with the large test as follows: Through the feed funnel introduce pulp of a consistency of $3\frac{1}{2}$ parts fluid to 1 part solids, until the large containing tube is filled to 5 ft. Since in a 5-ft. column of $3\frac{1}{2}$ to 1 pulp, $\frac{1}{2}$ part water is equivalent to 7.65 in., it follows that this particular pulp must settle a distance greater than 7.65 in. before it reaches the critical point. Therefore, 1 ft. of feed with a consistency of say 7 parts fluid to 1 part solids, may be added to bring the pulp surface up to cock D6. After 1 hr., test the settling by opening cock D5 to ascertain if clear fluid can be withdrawn. Close cock D5, open cock D6 and pour feed in through the funnel until a murky overflow

occurs at cock *D6*. Stop immediately the outflow at *D6* by closing the cock. Allow another hour to elapse and test as before. Repeat this at hourly intervals until it is found that the pulp level has risen to cock *D5*, as determined by pulp flowing out at this cock. When this stage is reached, either the feed must be decreased or thickened pulp must be removed from the cock at the bottom. Next remove 6 in. of thickened pulp through the bottom cock. This pulp will be sandy and its specific gravity need not be determined. Feed may now be added at the original rate until pulp again appears at cock *D5*, when it becomes necessary to remove thickened pulp from the discharge cock. The amount of feed added should be ascertained by measuring the volume of both the overflow and discharge. Continue the process as described until the specific gravity of the discharge is uniform and the sand content normal. The feed pipe should extend slightly below the cock at the top of the zone tested and the surface of the pulp should never be permitted to fall below the mouth of the feed pipe, for, should it do so, it is not definitely known that the maximum feed is being given. If the pulp in the discharge is not as dense as desired, reduce the feed and continue the test. After equilibrium has been reached, the capacity per square foot per hour is computed. Further tests with various depths in the experimental apparatus will give the necessary data for choosing the depth of tank required. To the depth of thickening zone determined by the experimental apparatus must be added the additional necessary depth for storage, feed, etc., as previously explained. The series of tests for maximum capacity in the glass cylinder will greatly assist the operator in making the large tests.

REMARKS CONCERNING SETTLING TESTS

When we asked certain operators to make settling tests to prove the value of the formula $C = \frac{62.35R}{F - D}$ we were not familiar with the erratic behavior of certain thick pulps containing large proportions of lime, nor with the importance of small changes in temperature. We therefore limited the ratings to single readings on each test and did not mention the effect of temperature. It is probable, in the few cases where a fairly close check between actual and indicated capacity is not shown when the thickeners were being worked to the limit of their capacity, that the discrepancy is due to one or the other of these causes.

In our earlier work the pulp samples tested were built up to give the desired consistency by combining the discharge and feed, as previously described. Subsequently it was found more convenient to start with a sufficient measured quantity of feed pulp and use the same pulp throughout the whole series of tests for maximum capacity, so each pulp sample

is made up by decanting measured quantities of clear liquid and mixing the remaining pulp well before taking the sample for each test. A thorough mixture is very important, especially with tests which are to be settled for density. With thin pulps the settling rate will not be altered by the sand; for instance, if the sample has less than its proportion of coarse sand, the settling rate remains unaffected. The amount of electrolyte should be the same throughout all the tests and the same as that to be used in practice. A high proportion of lime almost invariably causes a somewhat thinner discharge from the thickener. In making tests for compression and final thickening in low cylinders, the clear overlying liquid should be removed in order to keep the surface of the liquid down near the pulp surface.

Before increasing the proportion of electrolyte in mill practice to increase the settling rate, the series of tests for maximum capacities should be made and carefully analyzed to ascertain if the increase in the electrolyte will prove advantageous in the zone of the pulp the settling rate of which is limiting the capacity of the tank.

The temperature of tests must be uniform and the same as will be maintained in practice. High temperature increases the settling rates in most pulps and it is not rare to find the rate altered from 30 per cent. to 50 per cent. by a few degrees change of temperature. The condition of the pulp tested is of utmost importance. Samples dried in the air or by fire may be ruined for testing. Many colloids when dried become set, that is, lose their plasticity. On the other hand, ore, as it comes from the mine, may not be in a condition to test. A fresh ore, ground to 200 mesh, settled to 40 per cent. moisture in an hour and required less than 2 sq. ft. of settling area per ton per day; after 5 hr. in contact with cyanide solution containing lime, it required 3 sq. ft. of settling area and settled to 50 per cent. moisture in 1 hr.; after 24 hr., it required 5 sq. ft. of settling area and 20 hr. to settle to 60 per cent. moisture; after 20 hr. no further change occurred. A sample of the same ore, ground wet and left in a sack for a week, required 10 sq. ft. of settling area and settled to 66 per cent. moisture after 24 hr. It is therefore obvious that pulp should have precisely the same treatment before and during testing that it will in practice.

It is not within the scope of this paper to discuss the action of electrolyte, but we desire to call attention to the fact that when the use of an electrolyte is desirable, frequently the most expensive electrolyte is cheaper to use than a less expensive one. Some of the electrolytes available for use in commercial practice are lime, caustic soda, alum, iron sulphate, permanganate of potash, sulphuric acid and calcium sulphate. At times the addition of a small proportion of a colloid may be more effective in aiding settling than the use of an electrolyte. We have had called to our attention a case in clarifying dirty water

where 0.02 lb. of glue per ton was much more effective than $1\frac{1}{2}$ lb. of lime per ton.

Acknowledgment

We are indebted to the following gentlemen who had their local staffs make the tests recorded in curves, Figs. 4 to 13, inclusive: Chas. A. Chase, of the Liberty Bell Mining Co.; A. H. Jones, of the Belmont Milling Co.; Thomas B. Crowe, of the Portland Gold Mining Co.; R. B. Watson, of the Nipissing Mining Co.; E. M. Gleim, of the Presidio Mining Co.; P. A. Robbins, of the Hollinger Gold Mine, Alvin Carpenter, of the West End Consolidated Co., Allan J. Clark, of the Homestake Mining Co., and A. L. Blomfield, of the Golden Cycle Mining Co. We are also indebted to the Empire Co. and the Liberty Bell company for pulp samples, and to J. M. Tippet, L. B. Eames, and others, for making tests not recorded in the paper.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

BY HENRY PHELPS HOWLAND,* SOUTH CHICAGO, ILL.

INTRODUCTION

DURING the last decade no topic has created more interest or received more thought among blast-furnace men than *coke*.

One reason for this is, undoubtedly, the remarkable increase in the use of byproduct coke. Formerly our coke was made at a distance from the furnaces and by a distinct organization. There was little coöperation or development possible under these conditions.

Today the situation is greatly changed. Much of our coke is made by the organization operating the furnaces and the coke oven is operated with the primary object of obtaining the best results in the blast furnace.

The effect of changing the method of coke manufacture, even in some minor detail, is often felt almost immediately at the blast furnace. There naturally results one of the most valuable assets of the byproduct coke oven, *i.e.*, the possibility of increased blast-furnace efficiency due to close observance and correct operation of the ovens. The results are evident, but it often is difficult correctly to account for them.

In attacking the solution of a problem of as much importance as why one coke is better than another, there are facts which should be kept clearly in mind. The author hopes, in this paper, to bring out some points relative to the use of carbon in the blast furnace which will be of value in solving this question.

GRUNER'S IDEAL WORKING

Years ago Gruner made a statement regarding the use of carbon in the blast furnace, which has been very generally accepted by metallurgical and blast-furnace men. I quote this theory as given by Prof. Richards on page 248 of *Metallurgical Calculations*:

"From the standpoint of the generation of the maximum quantity of heat in the furnace, Gruner was right in formulating his dictum of the ideal working of a blast furnace, *viz.*: All the carbon burnt in the furnace should be first oxidized at the tuyères to CO and all reduction of oxides above the tuyères should be caused by CO, which thus becomes CO₂."

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TABLE I

Furnace No.	Month Year	Lb. Coke per Ton Iron	Tons Iron per Day	Carbon in Coke, Per Cent.	Kind of Coke		Gas Analysis—Per Cent. by Volume							Pounds Gas Carbon to			Calculated Quantities per Ton Iron			Pounds Carbon per Ton Iron			Per Cent. Carbon Burned at Tuyères				
					Method of Manufacture	Operation	CO ₂	CO	CH ₄	CO ₂ +CO+CH ₄	N	H	Cu. ft. Gas 62° F.							Cu. ft. Air 62° F.	Cu. ft. Air per Lb. Coke	Total Charged	Gasified in Furnace	Gasified at Tuyères	Per Cent. Total Carbon	Per Cent. Gasified Carbon	
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22						
1	1/21	2,615	301	86.3	BH	Stonega	10.3	27.6	0.2	38.1	59.8	2.1	2,251	187,200	141,600	54.2	2,254	2,110	1,868	82.8	88.6						
2	3/2	2,551	272	84.4	BP	Solva	11.3	26.9	0.6	38.8	59.0	2.2	2,182	178,200	132,800	52.0	2,153	2,049	1,751	81.4	85.6						
3	3/2	2,472	482	86.1	BH	Conn.	12.3	25.8	0.0	38.1	58.4	3.5	2,123	177,300	131,000	53.0	2,128	1,996	1,728	81.2	86.6						
4	1/24	2,247	450	87.1	BH	Conn.	11.8	26.2	0.1	38.1	58.5	3.4	1,978	164,440	121,600	54.2	1,957	1,846	1,605	82.0	87.0						
5	3/4	2,198	499	86.9	BH	Conn.	12.5	26.8	0.3	39.6	58.0	2.4	1,933	154,600	113,200	51.6	1,908	1,810	1,494	78.7	82.6						
6	3/4	2,123	541	88.3	BH	Conn.	12.8	26.0	0.2	39.0	58.0	3.0	1,905	154,800	113,500	53.4	1,875	1,764	1,498	79.8	84.9						
7	1/23	2,115	360	84.3	BP	Solva	14.2	25.3	0.0	39.5	57.9	2.6	1,826	146,500	108,200	51.2	1,782	1,683	1,427	80.1	84.8						
8	1/24	1,996	490	86.3	BP	Koppers	13.8	26.0	0.2	40.0	57.4	2.6	1,713	135,600	98,400	49.3	1,722	1,611	1,298	75.4	80.6						
9	3/4	1,936	376	85.7	BP	Solva	14.6	24.7		39.3	57.4	3.3	1,689	137,000	98,900	51.1	1,659	1,557	1,305	78.8	83.7						
10	3/4	1,905	393	88.7	BP	Solva	14.5	26.0		40.5	56.4	3.1	1,700	132,900	94,900	49.8	1,690	1,576	1,252	74.1	79.5						
11	1/25	1,901	517	85.5	BP	Koppers	13.6	24.9	0.2	38.5	58.0	3.2	1,610	132,400	97,000	51.0	1,625	1,524	1,280	78.8	84.1						
12	1/24	1,863	504	86.6	BP	Koppers	14.0	25.3	0.2	39.5	57.4	2.6	1,602	128,500	90,500	50.0	1,614	1,513	1,280	76.2	81.3						
13	3/4	1,780	426	84.9	BP	Koppers	14.2	26.0		40.2	56.2	3.6	1,519	120,000	85,200	48.0	1,511	1,414	1,124	74.4	79.5						
14	1/25	1,742	503	84.6	BP	Koppers	15.4	22.8	1.2	39.4	57.1	3.7	1,480	119,000	85,925	49.3	1,474	1,382	1,133	76.9	82.0						
15	1/24	1,716	542	87.1	BH	Benham	15.7	22.8		38.5	57.8	3.7	1,505	123,800	93,200	54.2	1,494	1,396	1,194	80.0	87.0						
16	1/25	1,715	585	84.6	BP	Koppers	15.4	22.6	1.4	39.4	57.1	3.5	1,486	117,300	84,530	49.3	1,451	1,357	1,114	76.7	82.2						
17	3/4	1,702	543	87.5	BP	Koppers	14.2	25.2	0.2	39.6	57.3	3.1	1,476	127,500	85,600	50.3	1,490	1,388	1,130	75.9	81.5						
18	3/4	1,699	572	87.0	BP	Koppers	14.9	23.5	0.2	38.6	57.3	3.1	1,472	120,800	87,600	51.6	1,479	1,387	1,155	78.2	83.4						
19	3/4	1,673	580	88.6	BH	Benham	15.1	23.6		38.7	58.6	2.8	1,477	121,000	89,600	53.4	1,482	1,384	1,182	79.9	85.0						
20	3/4	1,658	590	88.3	BH	Benham	15.5	22.9	0.1	38.5	58.7	2.8	1,467	120,600	89,600	54.0	1,464	1,366	1,182	80.8	86.5						
21	3/4	1,636	442	89.5	BP	Koppers	14.2	25.2	0.2	39.6	57.6	2.8	1,464	117,100	85,200	52.0	1,463	1,369	1,124	76.8	82.1						
22	3/4	1,635	593	88.5	BH	Benham	15.5	23.3	0.0	38.8	58.3	2.9	1,434	117,000	85,200	52.1	1,447	1,349	1,124	77.7	83.4						
23	3/4	1,624	592	87.3	BH	Benham	14.8	23.6	0.2	38.6	58.3	3.1	1,401	115,000	84,800	52.2	1,417	1,317	1,118	79.0	85.0						
24	3/4	1,623	457	89.6	BP	Koppers	15.3	24.8	0.2	40.3	57.0	2.7	1,453	114,700	82,600	50.9	1,454	1,360	1,090	75.0	80.2						
25	3/4	1,589	608	88.3	BH	Benham	15.3	23.5	0.0	38.8	58.2	3.0	1,360	103,400	83,300	52.4	1,403	1,307	1,100	78.5	84.2						
26	3/4	1,584	466	89.2	BP	Koppers	15.7	24.6	0.2	40.5	56.9	2.6	1,423	111,300	80,100	50.6	1,413	1,324	1,057	74.8	79.9						

TABLE II. (Data Additional to that of Table I)

Fur- nace No.	Dimensions				Capacity Cu. Ft. Tuyères to Stock Line	No. Tuyères	Date Last Blown In	Per Cent. Silicon in Iron	Per Cent. Carbon				Per Cent. Moisture in Coke	Pounds per Ton Iron		Engine Room Re- ports		Per Cent. Coke Screened Out			
	Heights	Diameter							Iron	Stone	Flue Dust	Coke Dried		Coke as Re- g- ist- ered	Stone Used	Flue Dust Made	Cu. Ft. Air per Min.		Temp. Air to Tubs	Grains Mois- ture	
		Total	Tuyères to Top Booth	Hearth Booth																	Stock Line
1	84'	13'4 1/4"	13'	19'3"	14'	11,060	8	9-09	1.62	4.0	11.8	25.0	88.7	80.0	1,192	225	30,040	34°	1.83	0.5	
2	80'	11'	11'0"	16'	12'	9,200	8	9-09	1.59	4.0	11.6	5.4	88.5	80.0	1,148	275	25,295			0.75	
3	89'	13'10"	15'6"	20'6"	16'	18,500	12	9-10	2.13	4.0	12.0	11.7	88.1		1,144	370	45,560	62°	1.55		
4	85'	13'0"	16'0"	22'0"	16'	19,500	12		1.56	4.0	12.0	8.2	89.8	82.0	1,097	461	44,213	82°		0.56	
5	85'	13'0"	16'0"	22'0"	16'	19,500	12		1.40	4.0	12.0	8.2	89.4	82.0	1,003	388	45,839	92°	6.44	0.39	
6	89'	13'10"	15'6"	20'6"	16'	18,500	12	9-10	1.93	4.0	11.7	12.0	90.7	80.0	1,208	180	46,090	52°	3.15	0.5	
7	80'	12'11"	12'0"	18'6"	13'6"	12,375	8	9-12	1.53	4.0	12.05	11.6	89.1	81.5	1,190	116	32,390	70	4.28	1.5	
8	94'	11'7"	17'0"	21'3"	16'3"	30,666	12	2-14	1.03	4.2	11.75	9.0	88.5		870	140	39,210	38	1.05		
9	80'	11'11"	13'6"	18'6"	14'0"	12,459	8	5-15	0.94	4.1	12.05	4.8	89.4	84.0	1,101	224	30,799	90	5.9	0.3	
10	80'	11'11"	13'6"	18'6"	14'0"	12,459	8	5-15	1.25	4.4	12.0	8.5	91.6	84.0	1,042	192	31,120	85	6.6	0.3	
11	Same as	No. 8							0.90	4.25	11.93	11.45	88.3		719	140	36,824	37	2.58		
12	Same as	No. 8							0.95	4.20	11.75	7.55	88.85		755	98	33,220	54	2.04		
13	78'	19'10"	14'	19'6"	15'0"	15,584	9	3-14	1.26	4.00	11.70		88.00		897	60	29,670	62			
14	85'	14'3"	15'	20'6"	15'5"		12	1-15	1.00	4.00	12.00	2.00	89.1	87.0	828	135	36,368	60	4.0	1.7	
15	90'	14'	16'6"	21'6"	16'0"	20,222	12	9-14	1.62	4.00	12.00	10.0	92.0	80.0	912	89	35,385	65		3.2	
16	90'	14'3"	16'6"	21'4"	16'6"		12	9-13	0.89	4.00	12.00	2.1	89.1	87.0	833	200	42,765	60	3.9	1.6	
17	90'	14'6"	17'8"	22'0"	17'0"	20,630	10	2-14	1.23	4.20	11.75	6.2	89.5		762	95	42,950	49	2.5		
18	Same as	No. 17							1.35	4.20	11.75	8.25	89.0		804	82	43,970	75°	5.5		
19	Same as	No. 15							1.56	4.00	11.95	9.5	93.3	80.0	1,85	780	94	35,615	62°	2.2	3.8
20	Same as	No. 15							1.41	4.00	11.9	9.1	92.6	80.0	2.23	849	90	35,820	63°	1.7	2.8
21	Same as	No. 13							1.36	4.1	12.0	4.8	91.8		792	31	30,050	70	3.8		
22	Same as	No. 15							1.62	4.0	11.9	8.0	92.2	80.0	1.4	709	99	35,210	56	3.8	3.0
23	Same as	No. 15							1.48	4.0	11.7	8.8	91.2	80.0	1.6	714	125	35,680	62	1.8	3.0
24	Same as	No. 13							1.19	4.1	12.0	6.9	91.9		775		29,910	75	5.0		
25	Same as	No. 15							1.67	4.0	11.7	7.1	92.3	80.0	1.5	712	94	35,570	62	3.4	3.3
26	Same as	No. 13							1.20	4.1	12.0	7.6	91.5		828		30,210	80	6.4		

Furnaces No. 24 and No. 26 Used More Flue Dust than Produced

This quotation shows clearly that Gruner was speaking from the standpoint of the heat development in the blast furnace. The blast furnace and metallurgical world in general have corrupted this theory and applied it to fuel consumption. We have come to believe that a low-coke furnace gets a larger percentage of coke down to the tuyères than a high-coke furnace.

Prof. Richards has well stated the commonly accepted view on the following page of the same book where he says:

"This proportion, or percentage, will not necessarily express how efficiently the furnace is running as regards fuel used per unit of iron made, but it will tell what proportion of the calorific power of the fuel used is being generated at the tuyères and in possibly *nine cases out of ten* this proportion indicates the general efficiency of the furnace as regards fuel consumption."

The following example illustrates this theory:

Take a furnace using 2,100 lb. of coke of 88 per cent. carbon, or 1,848 lb. of carbon per ton of iron: Assume that 100 lb. of this carbon is not gasified, but is used to supply the carbon for the iron and flue dust, thus leaving 1,748 lb. that could be burned at the tuyères. If we find that 85 per cent. of this, or 1,486 lb., was burned at the tuyères we must then say that this furnace is 85 per cent. efficient as regards fuel consumption. This line of reasoning would lead us to believe that when we reached 100 per cent. efficiency the carbon gasified at the tuyères would be 1,486 lb. and that would be all that was gasified. We would then require in this perfect furnace $1,486 \div 0.88 = 1,699$ lb. of coke or

$$\frac{1,586}{0.88} = 1,800 \text{ lb. of coke.}$$

Again, when a furnace was using 2,500 lb. of coke the efficiency would be

$$\frac{1,486}{(2,500) 0.88} \div 100 = \frac{1,486}{2,100} = 70.8 \text{ per cent.}$$

and only 70.8 per cent. of the carbon should be burned at the tuyères.

The only reason for the belief in this commonly accepted theory must be that no one has ever made any calculations along this line, on furnaces of high- and low-coke consumption.

CALCULATIONS SHOWING PERCENTAGE OF CARBON GASIFIED AT TUYÈRES

Somewhat over a year ago our low-coke furnace was found to be gasifying no greater percentage of coke at the tuyères than a previous furnace using 2,100 lb. of coke. This fact being proved for our individual furnaces, it seemed well to compare the results with those on other furnaces. With this in view, data from other plants were obtained and

the results of the calculations are shown in Table I. The method of calculation is as follows:

All figures are based upon the ton (2,240) of iron as a unit. All volumes are corrected to 62°F. The air blown is assumed to be dry at 62°F. which, of course, while not exactly true, is assumed thus to simplify the problem.

The total carbon charged in the coke minus the carbon in the pig iron, and the carbon in the flue dust, leaves as a remainder the carbon that is gasified and can leave the furnace only in the form of gas. The carbon in the limestone can also leave only in the furnace gas.

The sum of these amounts of carbon gives us the pounds of carbon in the furnace gas that can appear only in the form of CO, CO₂ and CH₄. The pounds of carbon in 1 cu. ft. of either CO, CO₂ or CH₄ is the same, due to Avogadro's law. Hence, having the total pounds of carbon going to the gas, we obtain the cubic feet of CO, CO₂ and CH₄.

The gas analysis shows what per cent. of the total gas these carbon gases constitute and thus enables us to obtain the cubic feet of gas per ton of iron. The gas analysis also gives the percentage of nitrogen and thus we arrive at the cubic feet of nitrogen per ton of iron, and as this can come only from the air, we determine the air actually delivered to the furnace per ton of iron. Dividing this latter figure by 75.8¹ gives us the pounds of carbon burned at the tuyères.

This can then be expressed as may be desired, either in per cent. of total carbon charged or of carbon gasified in the furnace. The method is outlined in a specific case as follows:

Furnace No. 1. (Plant) Wisconsin Steel Co. (Date) October, 1914

1. Tons per day, 542.
2. Coke, 1,716 lb.
3. Limestone, 912 lb.
4. Flue dust, 89 lb.
5. Carbon in coke, 87.08 per cent.
6. Carbon in limestone, 12 per cent.
7. Carbon in flue dust, 10 per cent.
8. Carbon in Fe, 4 per cent.
9. Gas, CO₂, 15.7 per cent.
10. Gas, CO, 22.8 per cent.
11. Gas, per cent. CH₄, 0.
12. Gas, N₂, 57.8 per cent.
13. Cu. ft. air per minute = 35,385.
14. Temp. of air to engines, 65°F.
15. Time lost, minutes per day, 21.
16. Pounds carbon to gas = (2)(5) + (6)(3) - (2,240)(8) + (4)(7) = 1,504.5.
17. Per cent. carbon gases in gas = (9) + (10) + (11) = 38.5.

¹ 1 cubic foot of air at 62°F. to burn 1 lb. of C to CO.

18. Pounds carbon gasified (16) - (3)(6) = 1,395.5.
19. Cubic feet gas at 62°F. = $\frac{(18)}{(17)} \times 31.69 = 123,848$.
20. Cubic feet actual air at 62°F. = $\frac{(16)}{(12)}$ per cent. N 40.06 = 90,497.
21. Pounds carbon gasified at tuyères, $\frac{(20)}{75.8} = 1,194$.
22. Carbon gasified at tuyères per cent. gasified carbon = 85.7 per cent.
23. Carbon gasified at tuyères per cent. total carbon = 80.0 per cent.
24. Cubic feet of air per pound of coke actual = 52.8 at 62°F.
25. Cubic feet of air per pound of coke, engine reports = 53.1 at 62°F.
26. Cubic feet of air, engine room reports $\frac{(13)(1,440 - (15)521)}{(1)(459 + (14))} = 92,200$ at 62°F.
27. Blowing efficiency = $\frac{(20)}{(26)} = 98.2$ per cent.

Aside from the fact that there may be some nitrogen from the coke and that we have assumed the air to be dry at 62°F., the method is exact. In my opinion, neither of these points is of enough value to counteract the complications introduced by using them. It is realized, of course, that the value of such a table depends primarily upon the accuracy of the data upon which the calculations are based. When the data were requested from the several different plants, the desirability of having it accurate was given the utmost emphasis.

COMMENTS ON DATA

The greatest stumbling block was gas analysis, and I am becoming more and more convinced that the average analysis of blast-furnace gas is almost worthless for calculation purposes. The reason for this lies primarily in its sampling, which, to obtain correct results, needs to be closely followed up by someone who realizes its importance and will see that the sampling is correctly done.

We have followed up this matter quite persistently at our plant for several years. There are two things which, if watched, will, generally speaking, show the accuracy of gas analysis:

First: Per cent. of hydrogen.

Second: The sum of the carbon gases.

Hydrogen.—This figure should be different for different periods of the year. The moisture in the ores and coke gives about 2 per cent. hydrogen in the gas. Five grains of moisture in the blast means about 1 per cent. hydrogen in the gas. Thus, in the winter months, our hydrogen should be about 2.3 per cent. and increase gradually up to 3 per cent. and, possibly, more in the more humid days. Of course, with a dry blast plant this figure would remain practically a constant at, say, 2.0 per cent.

Regarding the sum of the carbon gases, there is no reason, with a given condition of furnace operation, why this should not remain a practical

constant. It certainly should not vary much from day to day and should never be much below 38 per cent. or much above 40.0 per cent.

In some of the data sent me, the hydrogen was as low as 1 per cent. and the sum of the carbon gases varied from 36 per cent. to 41 per cent.

By careful observance of these two points one can be sure of getting as accurate analysis as is practicable to obtain. A gas analysis incorrect as to the sum of the carbon gases will make impossible correct determinations of the wind blown. The degree to which this error affects the wind blown may be illustrated as follows:

Referring to our general formula for air

$$A = \frac{\text{lb. of carbon to gas}}{\text{per cent. of carbon gases}} \text{ per cent. N (40.06)}$$

We will assume three different gas analyses, only one of which is correct:

(1) CO₂, 15 per cent.; CO, 24 per cent.; CH₄, 00; H, 2.5 per cent.; N, 58.5 per cent.

(2) CO₂, 15.5 per cent.; CO, 24.5 per cent.; CH₄, 00; H, 2.5 per cent.; N, 57.5 per cent.

(3) CO₂, 15 per cent.; CO, 26 per cent.; CH₄, 00; H, 2.5 per cent.; N, 56.5 per cent.

The sum of the carbon gas is 39 per cent., 40 per cent., and 41 per cent. respectively, the 39 per cent. being assumed as correct. Representing all the constants in our equation by K, the equation may be written

$$A = K \frac{\text{per cent. N}}{\text{per cent. sum of carbon gases}} = K \frac{100 - \text{per cent. H} - \text{per cent. N}}{\text{per cent. sum of carbon gases}}$$

$$(1) A = K \frac{0.585}{0.39} = K(1.50) \text{—Correct analysis used.}$$

$$(2) A = K \frac{0.575}{0.40} = K(1.44) \text{—Incorrect analysis used.}$$

$$(3) A = K \frac{0.565}{0.41} = K(1.39) \text{—Incorrect analysis used.}$$

This indicates that for each 1 per cent. error in reporting the sum of the carbon gas analyses, there will result 4 per cent. error in the calculated wind.

The column No. 5 in Table I headed "Carbon in Coke" may need a word of explanation. The way this figure is obtained at various plants differs. For example, at our plant while we screen out from our pockets about 3 per cent. of the coke, the coke figure per ton of iron is the total coke unloaded from cars into the pockets, divided by the tons of iron produced. In this case the car weights are those of the coke plant located several hundreds of miles from the furnace. At some of the by-product coke plants of the Chicago district the coke is screened at the

ovens and the furnace is charged only with the screened coke. The method of arriving at the carbon in the coke is illustrated as follows:

Coke.—Analysis, dried: Carbon 92 per cent. Moisture 1.5 per cent.

Screenings 3 per cent. Carbon in screenings 80 per cent.:

$(0.92 \times 0.985) - (0.80 \times 0.03) = 88.4$ per cent. = carbon in coke.

If the coke unloaded into the pockets divided by the production gives a coke consumption of 1,700 lb. then $1,700 \times 0.884 = 1,503$ lb. would give the carbon charged.

If it is not the practice to charge the furnace with the screenings then the coke consumption would be $1,700 \times 0.97 = 1,649$ lb. and the carbon in the coke $(0.92 \times 0.985) = 90.62$ per cent. The carbon charged would be $1,649 \times 0.9062 = 1,494$.

This latter figure is lower than the correct figure because it is really incorrect to credit the screenings as a certain per cent. of the coke because the carbon of the screenings is lower than the carbon of the coke.

I am confident that on the whole the data used in these calculations are as accurate as could be obtained. Any blast-furnace man who is inclined to question the accuracy of the results obtained has only to apply the method to his own furnaces.

CONCLUSIONS FROM TABLE I

Table I shows clearly that, at least in these furnaces, there is no law governing the relation between coke consumption and the per cent. of the carbon which is burned at the tuyères. For example, compare furnaces Nos. 6 and 18:

No. 6. Coke consumption, 2,198 per cent. Gasified at tuyères
= 82.6.

No. 18. Coke consumption, 1,699 per cent. Gasified at tuyères
= 83.4.

There is a difference of 500 lb. of coke per ton of iron, with practically no difference in the per cent. burned at the tuyères.

Starting with the high-coke furnaces at the top and glancing down the table, we find several such instances. Many of the furnaces toward the bottom of the table might be classed as ideal, from the standpoint of coke consumption, yet none of them approach the point of burning 100 per cent. of the carbon at the tuyères.

The conclusion is plain, therefore, that none of the above furnaces (furnaces which I believe to be fairly representative of American blast-furnace practice using Mesaba ores) are working on the basis commonly called "Gruner's Ideal."

The question very naturally arises whether any blast furnace does or ever has gasified 100 per cent. of its gasified carbon at the tuyères.

As far as the author is aware, no such instance based on accurate data has ever been cited. Furthermore, I believe that there is no reason why this condition should be desired.

CUBIC FEET OF AIR PER POUND OF COKE

Few figures in metallurgical writing are subject to more abuse in their quotation and use than that of cubic feet of air per pound of coke. We hear it quoted all the way from 45 to 70 and usually with no specification as regards temperature. We hear it used as a basis of comparison of blowing equipment and even to prove that Greenawalt sinter is of great value, as compared to some of our fine Mesaba ores.²

There are, of course, two values of this figure; the one almost universally used is the wind per pound of coke as quoted in our daily and monthly reports; the other, which is rarely used, the actual value as shown in Table I.

The first value, which we will speak of as the "approximate wind per pound of coke" of course varies greatly. The "approximate" value is generally obtained by multiplying the engine revolutions for 24 hr. by the air supposedly blown per revolution, and dividing this product by the number of pounds of coke supposedly charged during the 24 hr. This gives the usual "approximate wind per pound of coke" for 24 hr. No allowance being made for temperature, this figure cannot be quoted at any standard temperature.

It is only in recent years that any allowance has been made for variations in temperature. Many plants are at present using outside air and varying the revolutions of the engines with the thermometer. At many of these plants the wind per pound of coke is not reported at any standard temperature.

Another error is due to a varying coke unit. At the majority of plants the coke unit varies, as used from day to day, and at the end of the month it is found necessary to adjust the amount of coke charged to the furnace. The approximate wind per pound of coke for the month is, however, generally an average of the daily figure without any such correction being made.

The correct method of deriving this figure is, first of all, to know the temperature of the air entering the air tubs. The total cubic feet of air for the entire 24 hr. (based upon the engine displacement and allowing proper corrections for delays) corrected to 62°F. should be carried forward each day and added for a total for the month. By using this sum as a dividend, and the correct shipper's weight of coke

² J. E. Johnson: Increased Fuel Economy in Smelting Iron Ores, *Iron Trade Review*, vol. liv, p. 239 (1914).

for the month as a divisor, our quotient will be the "approximate air per pound of coke" delivered by the engines at 62°F.

The above method is the way to report our cubic feet of air per pound of coke as delivered by our engines and yet I believe that there are few plants figuring it as carefully as outlined above.

Aside from the two errors in method of calculation mentioned, there are the variables of air tubs, leaks in mains, valves and stoves, all of which tend to make the approximate wind per pound of coke unreliable.

REMARKS ON APPROXIMATE VALUE

Table III shows some values of this figure which are on record and which are fair samples of the way this figure is quoted. It may be but natural that furnace men have come to look upon much of the variation which has occurred in the quotation of this figure as due to conditions inside of the furnace.

In attempting to account for these variations we have assumed that they were due to the kind of coke, the way the coke was being burned, an easily reduced ore, or an ore difficult to reduce, etc.

When the facts in the case are realized, we find that practically all of the variation is due to our methods of figuring, variations in temperature, leaks in stoves, mains, engine efficiency, etc.

TABLE III.—*Variation in Quoted Figures of Wind per Pound of Coke*

Period	Number of Furnaces	Average		Individual Furnace Figures							
				Lowest Coke		Highest Coke		Lowest Air		Highest Air	
		Coke	Air per Lb. Coke	Coke	Air per Lb. Coke	Coke	Air per Lb. Coke	Air per Lb. Coke	Coke	Air per Lb. Coke	Coke
Year—1906.....	91	2,343	64.9	1,989	56.8	3,238	60.0	53.1	2,137	82.0	2,419
Year—1907.....	94	2,362	65.0	1,945	68.6	3,212	63.0	48.3	2,473	87.8	2,337
Month—1908...	41	2,321	59.4	1,994	71.6	3,378	52.2	50.5	2,337	71.6	1,994
Week—1913....	10	2,030	67.3	1,703	73.1	2,264	68.7	57.6	2,056	73.1	1,703
Month—1907...	11	2,464	72.6	2,245	85.6	2,996	90.4	60.8	2,396	90.4	2,996
Month—1908...	6	2,375	57.2	2,041	53.5	2,843	62.3	52.0	2,585	65.2	2,287

REMARKS ON ACTUAL VALUE

Turning now to our other value, the actual wind per pound of coke, we find that there are only two causes for variation:

First: The carbon in the coke.

Second: The per cent. of carbon which reaches the tuyères.

We have seen that it takes 75.8 cu. ft. of air at 62°F. to burn 1 lb. of carbon at the tuyères.

Let us assume for illustration a furnace using 2,000 lb. of coke per ton of iron; the coke is charged having 87 per cent. carbon and 100 lb. of carbon being used to supply the iron and flue dust.

It follows that if every pound of coke that went into the top of the furnace reached the tuyères and was burned to CO, the maximum wind possible per pound of coke would be:

$$75.8 \times \frac{87}{100} = 65.95 \text{ cu. ft. air per pound of coke.}$$

This amount of air never could be used, simply because 100 lb. of the carbon was used for iron and flue dust, thus

$$2,000 \times 0.87 = 1,740 \text{ lb. of carbon charged,}$$

$$1,740 - 100 = 1,640 \text{ lb. of carbon gasified,}$$

$$\frac{1,640}{1,740} = 94.3 \text{ per cent. of total carbon gasified in furnace.}$$

We therefore see that in reality the maximum air that can be used is:

$$75.8 \times \frac{87}{100} \times \frac{94.3}{100} = 62.2 \text{ cu. ft. air per pound coke.}$$

This furnace would be working along the lines of Gruner's Ideal furnace.

We have seen from Table I that instead of 94.3 per cent. of the total carbon reaching the tuyères, the figure is much more nearly 80 per cent. Hence, the correct figure becomes:

$$75.8 \times \frac{87}{100} \times \frac{80}{100} = 52.76 \text{ cu. ft. of air per pound of coke.}$$

This figure will be a little higher as the coke consumption increases, due to the fact that the carbon necessary for the iron and flue dust is practically the same in pounds, but naturally a lower percentage of the total, thus leaving a higher percentage which can be gasified.

RELATION BETWEEN WIND, COKE AND PRODUCTION

A clear conception of the fact that the actual wind per pound of coke is a practical constant should tend to emphasize in our minds the value of low wind where we are seeking primarily low coke consumption.

Tables IV and V are inserted and Fig. 1 plotted with the idea of emphasizing this point.

These are based upon the following empirical assumptions which, aside from the blowing efficiency, are believed to be practically exact for nearly all furnaces:

First: 52.5 cu. ft. of dry air at 62°F. per pound of coke.

Second: Engines delivering blast 1,420 min. out of 24 hr., thus assuming a delay of 20 min.

Third: Blowing efficiency 100 per cent.

Let us assume that we have a furnace that should make 540 tons per day. By blowing various amounts of air we obtain the following results:

Cubic Feet Air per Minute	Coke Consumption	Tons per Day
34,000	1,700	541
36,000	1,800	540
38,000	1,900	540
40,000	2,000	540
42,000	2,100	541

On the other hand, if our coke consumption is to stay the same with the increased wind, then our tonnage will have to increase as follows:

Cubic Feet Air per Minute	Tons per Day	Coke Consumption
34,000	541	1,700
36,000	572	1,700
38,000	604	1,700
40,000	636	1,700
42,000	668	1,700

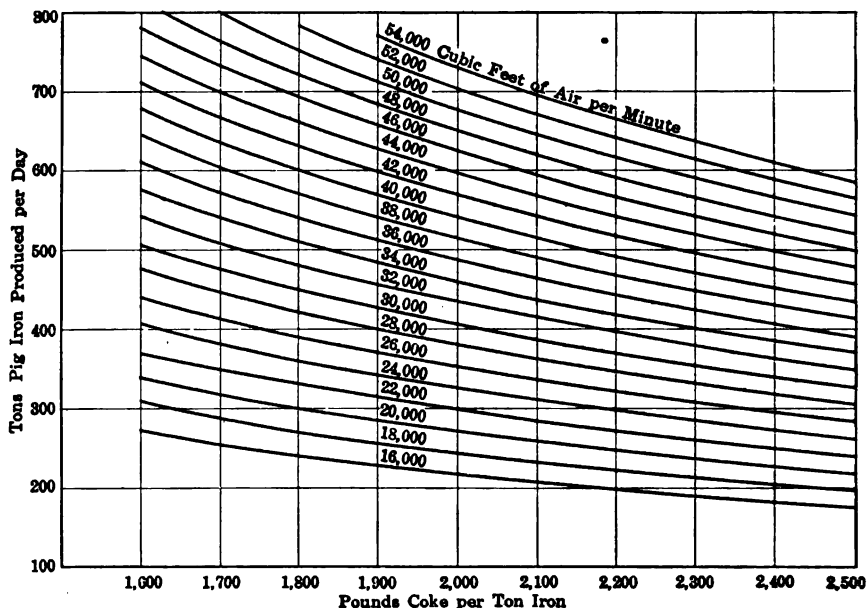


FIG. 1.

There could be little doubt as to which way the increased wind would work. As a matter of fact, it probably would not work out along either line exactly, but would result in both increased tonnage and coke, so that when we reached 42,000 cu. ft. of air per minute, the result might be about 568 tons and 2,000 coke consumption, an increase of 28 tons per day and 300 lb. of coke per ton of iron, due to increasing the wind from 34,000 to 42,000 cu. ft. of air per minute.

This table shows with reasonable accuracy the wind we are blowing per minute for any particular furnace. One who is operating a furnace

producing 542 tons per day on 2,100 lb. of coke is probably actually blowing about 42,000 cu. ft. of air at 62°F. If, according to engine reports, the figure is 45,000, then the blowing efficiency is $\frac{42,000}{45,000} = 93.4$ per cent.

These curves show us at once that it is practically impossible to obtain low-coke consumption unless we keep our wind low.

TABLE IV.—*Relation between Cubic Feet of Air per Minute, Tons of Iron Produced, and Coke per Ton of Iron, Assuming Wind per Pound Coke = 52.5 Cu. Ft. Blast Delivered to Furnace 1,420 Min. per Day*

Cu. Ft. Actual Air per Min. at 62°F.	16,000	18,000	20,000	22,000	24,000	26,000	28,000	30,000	32,000
Coke per Ton Iron									
1,600	271	304	338	372	406	440	474	507	541
1,700	255	287	318	350	382	414	446	477	509
1,800	240	270	300	330	360	390	420	450	480
1,900	228	256	285	314	342	371	399	428	456
2,000	217	243	271	298	325	352	379	406	433
2,100	206	232	258	284	310	336	362	387	413
2,200	197	221	246	271	296	321	345	370	396
2,300	188	212	235	258	282	305	329	352	376
2,400	180	203	226	248	271	294	316	339	361
2,500	173	195	216	238	259	281	303	325	348

TABLE V.—*Relation between Cubic Feet of Air per Minute, Tons of Iron Produced, and Coke per Ton of Iron, Assuming Wind per Pound Coke = 52.5 Cu. Ft. Blast Delivered to Furnace 1,420 Min. per Day*

Cu. Ft. Actual Air per Min. at 62°F.	34,000	36,000	38,000	40,000	42,000	44,000	46,000	48,000	50,000	52,000	54,000
Coke per Ton Iron											
1,600	575	608	642	677	711	744	778	812	846	879	913
1,700	541	572	605	637	669	701	732	764	796	828	859
1,800	510	540	570	600	630	660	690	720	750	780	811
1,900	484	512	541	570	599	627	655	684	713	741	768
2,000	460	487	514	541	569	595	623	649	677	704	730
2,100	438	463	489	515	542	568	593	618	645	670	695
2,200	418	442	467	493	517	542	566	590	615	640	664
2,300	400	423	447	471	495	518	541	565	589	612	635
2,400	383	406	429	451	474	496	519	541	564	586	609
2,500	368	389	411	433	455	477	498	519	542	563	584

ANALYSIS OF USES OF CARBON

Table I shows a wide range of furnaces considered from the standpoint of coke consumption. The furnace at the top of the list burned 1,868 lb. of carbon at the tuyères, whereas the best furnace burned only 1,057 lb. The work accomplished was approximately the same in both cases. We are not so much interested in the question why the high-coke furnace

took 1,868 lb. as we are in how the other furnace was able to get along with 1,057 lb.

It has occurred to me that one help toward solving this problem is to analyze as simply as possible the work done by the carbon in the furnace. I have therefore taken a furnace which, while not the lowest in the list, possibly serves better for comparison because of that fact—furnace No. 19, using 1,673 lb. of coke.

This furnace is No. 1 Furnace of the Wisconsin Steel Company for the month of February, 1915. The data are as follows:

Produced: 580 gross tons of iron per day.
 94 lb. of flue dust per ton of iron.
 Consumed: Coke 1,673 lb. per ton of iron.
 Stone 780 lb. per ton of iron.
 Scrap 75 lb. per ton of iron.

ANALYSIS

	Per Cent.
Iron: Silicon.....	1.560
Manganese.....	0.750
Phosphorus.....	0.075
Carbon.....	4.000
Sulphur.....	0.035
Iron.....	93.580
Coke—as charged.....	88.6 Carbon
Stone.....	12.0 Carbon
Flue dust.....	9.5 Carbon
Gas: CO ₂	15.1
CO.....	23.6
H.....	2.7
N.....	58.6
Ratio, CO/CO ₂	1.49

By the same methods used in calculating Table I, we obtain the following results:

Pounds carbon charged.....	1,482
Pounds carbon gasified in furnace.....	1,384
Pounds carbon gasified at tuyères.....	1,182
Cubic feet of gas at 62°F.....	120,950
Cubic feet of air at 62°F.....	89,600

	Pounds
Carbon pig iron.....	89.4
Carbon flue dust.....	8.9
Carbon in stone.....	98.3
	93.6
CO ₂ in stone.....	$93.6 \times \frac{44}{12} = 343 \text{ lb}$

The carbon in the stone does not, of course, interest us, except that it furnishes 343 lb. of CO_2 which either goes to the gas, or attacks the carbon of the coke.

The Fe per ton of pig iron is: $2,240 \times 0.9358 = 2,095$ lb.

The excess scrap furnishes Fe as follows: $75 \times 0.85 = 64$ lb.

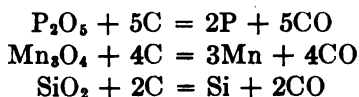
Thus leaving $2,095 - 64 = 2,031$ lb. of Fe to come from Fe_2O_3 .

This means

$$\frac{2,031}{0.70} = 2,901 \text{ lb. of } \text{Fe}_2\text{O}_3.$$

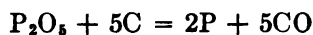
A certain amount of carbon is used in reducing the manganese, phosphorus and silicon.

The equations of these reactions are taken from Prof. Richard's book, quoted previously.



The atomic weights used in these equations are as follows:

Fe.....	56.0
H.....	1.008
O.....	16
C.....	12
P.....	31
Mn.....	55
Si.....	28



1 lb. of P requires $\frac{30}{31}$ lb. of C

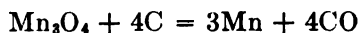
Per ton of iron we have—

$2,240 \times 0.00075 = 1.7$ lb. of phosphorus,

requiring $\frac{1.7 \times 30}{31} = 1.6$ lb. of C and produces $\frac{70}{31}$ lb. of CO.

producing $\frac{1.7 \times 70}{31} = 3.8$ lb. of CO

Manganese reaction—



1 lb. Mn requires $\frac{16}{55}$ lb. of C and produces $\frac{112}{165}$ lb. of CO.

Per ton of iron we have:

$$2,240 \times 0.0075 = 16.8 \text{ lb. of Mn}$$

requiring $\frac{16.8 \times 16}{55} = 4.88$ lb. of C,

producing $\frac{16.8 \times 112}{165} = 11.4$ lb. of CO.

Silicon reaction— $\text{SiO}_2 + 2\text{C} = \text{Si} + 2\text{CO}$

1 lb. of Si requires $\frac{24}{28}$ lb. of C. Produces $\frac{56}{28}$ lb. of CO.

Per ton of iron we have:

$$2,240 \times 0.0156 = 35 \text{ lb. of C}$$

requires $-35 \times \frac{6}{7} = 30$ lb. of C

produces $-35 \times 2 = 70$ lb. of CO

SUMMARY

	Carbon Required, Pounds	CO Produced, Pounds
Phosphorus reduction.....	1.6	3.8
Manganese reduction.....	4.9	15.0
Silicon reduction.....	30.0	70.0
Total.....	36.5	88.8

We originally had 1,384 lb. of carbon gasified in the furnace and we have used 36 lb. as shown above, leaving $(1,384 - 36) = 1,348$ lb. available for reduction of Fe_2O_3 .

Of this we know that 1,182 lb. is burned to CO at the tuyères, leaving 166 lb. of carbon which can be used in direct reduction. We will therefore have available for reduction 2,901 lb. of Fe_2O_3 , 166 lb. of solid carbon and $1,182 \times \frac{28}{12} = 2,758$ lb. of CO.

The amount of Fe_2O_3 we can reduce by means of these agents depends entirely upon how efficiently the work is done. We are told that there are a great many angles to this question of reduction. The temperatures and composition of the reducing gases may so interact upon one another that either oxidation or deoxidation may take place. There are many points involved in this question regarding which the author frankly admits his ignorance aside from what he reads in technical literature, and which he has no desire to discuss here.

Certain it is that our opinions on this topic change from time to time. Sir Lowthian Bell, upon whose researches most of our conclusions are based, remarks that while it might be possible, he doubts the probability of our ever being able to attain a ratio less than 2 to 1, by volume of CO to CO_2 in our top gases.

Our analysis of the top gas on our small hand-filled furnace of yesterday (Dec. 11, 1915) showed a ratio of 1.23.

I shall assume for the purpose of our discussion that the reduction of the Fe_2O_3 by either CO or C takes place under such conditions as to produce a ratio of 1 to 1 in the resulting gas. Any CO in excess of the amount required to reduce the ore under these assumptions must be considered as unused, and as passing to the gases as CO, thus, of course, raising the ratio in the top gases above that of the reducing gases.

In the same way we will have to assume that any solid carbon not necessary for reduction is dissolved by the CO_2 of the gas, thus raising the CO percentage in the top gas.

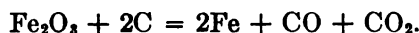
These empirical assumptions are made merely for the purpose of analyzing the work of the furnace, and as long as this point is kept clearly in mind, no confusion need arise.

The equation of reduction when written to meet these specifications will be as follows:

(A) *Indirect:*



(B) *Direct:*



It will be noticed that in the case of the "indirect" reduction we employ 6C to reduce the same amount of Fe_2O_3 that is accomplished by 2C in the equation of the "direct" reduction. According to this line of reasoning, direct reduction is three times as efficient, as regards carbon consumption, as is the indirect.

Returning now to the furnace under discussion, we can say that regardless of how the equations are written, we are sure of the following facts:

First.—2,901 lb. of Fe_2O_3 went into this furnace as ore and came out as 2,031 lb. of Fe in the iron and 870 lb. of oxygen in the gases.

Second.—1,384 lb. of carbon went into the furnace as solid carbon and came out as either CO or CO_2 in the gas.

Our problem, as previously stated, is: given 2,901 lb. of Fe_2O_3 to be reduced by means of 166 lb. of solid carbon, and 2,758 lb. of CO. We will make two assumptions:

Assumption A

That 166 lb. of carbon is used in direct reduction of the ore, the remaining ore being reduced by CO, all excess CO passing to the top gases.

The equation for direct reduction—



tells us that 166 lb. of C reduces $(166)6.667 = 1,107$ lb. of Fe_2O_3 and produces

$$(166)(1.167) = 194 \text{ lb. of CO}$$

$$(166)(1.833) = 304 \text{ lb. of CO}_2$$

We have left $2,901 - 1,107 = 1,794$ lb. of Fe_2O_3 which must be reduced indirectly. This requires $(1,794) \frac{168}{160} = 1,884$ lb. of CO for its reduction, and at the same time produces—

$$1,884 \times 0.50 = 942 \text{ lb. of CO}$$

and

$$1,884 \times \frac{11}{14} = 1,580 \text{ lb. of CO}_2$$

We have left—

$2,758 - 1,884 = 874$ lb of CO which passes to the gases without doing any reducing:

Summary—Assumption A

	Iron Produced	Carbon Used, Pounds	Pounds CO	Produced CO ₂
Si, Mn, and P reduction		36	89.0	0
Direct reduction of Fe_2O_3	775	166	194.0	305
Indirect reduction of Fe_2O_3	1,256	807	942.0	1,480
Unused CO.....		375	874.0	
Stone.....				343
Totals.....	2,031	1,384	2,099.0	2,128

Assumption B

Assume that all CO made at tuyères reduces ore and that the excess carbon is used up by CO_2 .

2,758 lb. of CO will reduce

$$2,758 \left(\frac{160}{168} \right) = 2,627 \text{ lb. of } \text{Fe}_2\text{O}_3$$

and produce

$$2,758 \times \frac{5}{10} = 1,379 \text{ lb. of CO}$$

and

$$2,758 \times \frac{11}{14} = 2,167 \text{ lb. of CO}_2$$

We have left $2,901 - 2,627 = 274$ lb. of Fe_2O_3 which must be reduced by direct reduction.

This requires—

$$274 \times \frac{3}{20} = 41 \text{ lb. of C}$$

produces

$$274 \times \frac{7}{40} = 48 \text{ lb. of CO}$$

and

$$274 \times \frac{11}{40} = 75 \text{ lb. of CO}_2$$

We have left $166 - 41 = 125$ lb. of carbon for which there is no way of accounting except upon the basis of its having been dissolved by CO_2 , thus:



125 lb. of C will dissolve

$$(125) \frac{(11)}{3} = 458 \text{ lb. of CO}_2$$

and produces

$$(125) \frac{14}{3} = 583 \text{ lb. of CO}$$

It should be noted that this furnace did not produce enough CO at the tuyères to reduce all the Fe_2O_3 .

Summary—Assumption B

	Iron Produced	Carbon Used, Pounds	Produced Pounds CO	Pounds CO ₂
Si, Mn, and P reduction.....		36	89	0
Direct reduction, Fe_2O_3	192	41	48	75
Indirect reduction.....	1,839	1,182	1,379	2,167
Stone.....				343
		1,259	1,516	2,585
Dissolved by CO_2		125	583	-458
Total.....	2,031	1,384	2,099	2,127

We thus see that in whichever way we assume the reduction to take place the ultimate gas analysis is the same.

The gas analysis obtained by analyzing the gas of this furnace was 23.6 per cent. CO and 15.1 per cent. CO_2 , and the cubic feet of gas was 120,950.

$$120,950 \times 0.236 = 28,544 \text{ cu. ft. of CO}$$

$$120,950 \times 0.151 = 18,213 \text{ cu. ft. of CO}_2$$

$$(28,544)(0.07363) = 2,102 \text{ lb. of CO}$$

$$(18,213)(0.1157) = 2,107 \text{ lb. of CO}_2$$

which shows that our data checks fairly well.

Assumption G

Let us suppose for the purpose of comparison that this furnace had consumed the same amount of coke but that the total of 1,348 lb. of carbon had all been burned to CO at the tuyères. Under this assumption the furnace would be working along the lines of Gruner's Ideal.

This would require $1,348 \times 75.8 = 102,178$ cu. ft. of air at 62° instead of 89,600 originally blown.

We will now produce $1,348 \times \frac{28}{12} = 3,145$ lb. of CO.

We have 2,901 lb. of ore to reduce which requires $(2,901) \frac{21}{20} = 3,042$ lb. of CO, and produces

$$3,042 \times \frac{5}{10} = 1,521 \text{ lb. of CO}$$

and $3,042 \times \frac{11}{14} = 2,387$ lb. of CO₂.

We have left 103 lb. of CO unused.*

Assumption G—Summary

	Iron Produced	Carbon Used, Pounds	Pounds to Gas	
			CO	CO ₂
Si, Mn, and P reduction.....		36	89	
Direct reduction, Fe ₂ O ₃		0	0	0
Indirect reduction, Fe ₂ O ₃	2,031	1,304	1,521	2,387
Unused		44	103	0
Dissolved by CO ₂		0	0	0
Stone				343
Total.....	2,031	1,384	1,713	2,730

Our gas analysis by volume will therefore become:

$$\text{CO} = \frac{1713}{0.07363} = 23,265 \text{ cu. ft. CO}$$

$$\text{CO}_2 = \frac{2730}{0.1157} = 23,595 \text{ cu. ft. CO}_2$$

$$\text{N} = (102,178)(0.791) = 80,823 \text{ cu. ft. of N.}$$

$$\text{CO} + \text{CO}_2 + \text{N} = 127,683 \text{ cu. ft.}$$

The hydrogen being 2.7 per cent. of the total, the above sum is 97.3 per cent. of the total gas, or the cubic feet of gas produced at 62° is—

$$\frac{127,683}{0.973} = 131,226.$$

The gas analysis is:

	Per Cent.
CO.....	17.7
CO ₂	18.0
H.....	2.7
N.....	61.6
	100.0

* Capable of reducing 44 lb. of Fe₂O₃.

The analysis of the working of this furnace under the three conditions outlined above, is shown in Table VI. In looking over this table we see that had this particular furnace operated as shown in the Column "G" (that is, according to Gruner's law) the net result to the furnace plant would be a distinct loss. While the furnace would have used the same amount of coke per ton of iron the B.t.u. available in the furnace gas is greatly lowered. The result shows that for each pound of carbon gasified in the furnace, we would have used 1,256 B.t.u. more in the furnace when operating as "G" than when operating as the furnace really did. As this furnace gasified 1,384 lb. of carbon per ton of iron, and made 580 tons of iron per day, the net loss would be $\frac{1,384 \times 580 \times 1,256}{11,000}$ = 91,600 lb. of coal in 24 hr., assuming coal to have a heat value as fired in the B.F. boiler of 11,000 B.t.u. per pound.

TABLE VI.—*Use of Carbon under Various Assumptions*

Per Ton Iron (2,240 Lb.)	Wisconsin Furnace No. 1		
	"A"	"B"	"G"
1. Coke, pounds.....	1,673	1,673	1,673
2. Carbon charged, pounds.....	1,482	1,482	1,482
3. Carbon gasified, pounds.....	1,384	1,384	1,384
4. Carbon reduction Mn. P, Si, pounds.....	36	36	36
5. Carbon direct reduction Fe ₂ O ₃ , pounds.....	166	41	0
6. Carbon indirect reduction Fe ₂ O ₃ , pounds.....	807	1,182	1,304
7. Carbon in CO not needed for reduction, pounds.....	375	0	44
8. Carbon dissolved by CO ₂ , pounds.....	0	125	0
9. Carbon gasified at tuyères, pounds.....	1,182	1,182	1,348
10. Carbon used in reducing Fe ₂ O ₃ , pounds.....	973	1,223	1,304
11. Cubic feet, air per pound, coke, 62°F.....	53.5	53.5	61.08
12. Analysis of Top Gas			
13. CO per cent. by volume, per cent.....	23.6	23.6	17.7
14. CO ₂ per cent. by volume, per cent.....	15.1	15.1	18.1
15. H per cent. by volume, per cent.....	2.7	2.7	2.7
16. N per cent. by volume, per cent.....	58.6	58.6	61.6
17. Total B.t.u. per cubic foot top gas at 62°.....	84.0	84.0	64.8
18. Cubic feet gas per ton iron.....	121,000	121,000	131,200
19. B.t.u. per pound C gasified.....	7,344	7,344	6,143
20. B.t.u. in CO per cubic foot top gas.....	76.6	76.6	57.4
21. B.t.u. in CO per pound carbon gasified.....	6,697	6,697	5,441
22. B.t.u. lost to top gas per pound carbon gasified.....			1,256

Comparing the gas analyses on these two furnaces, we see at once that they give no indication of the coke consumption, but only of the way the reduction takes place in the furnace. If we had been able to operate this furnace along the lines of "A" or "B" and produce a gas

with the same $\frac{\text{CO}}{\text{CO}_2}$ ratio as "G" the coke consumption would have been far below that shown in "G."

It is difficult for me to see how this furnace, as operated, would have been improved by endeavoring to operate it along Gruner's ideal, even assuming this to be a possibility.

J. E. Johnson, Jr.,³ has just finished a very complete analysis of the question of heat development in the blast furnace, and has found himself unable to depart from his original belief in the correctness of Gruner's ideal working. I have not as yet had time to give his article the study it deserves and it may be entirely possible that when I do, my ideas along this line will change.

In Mr. Johnson's article he has attempted to bring into line the results obtained on our No. 1 Furnace with those obtained by his figuring. In so doing, he has quoted our blast heat as 1,250°F. and the carbon in the coke as 93 per cent. The correct figures are 1,100° and 89 per cent.

His whole case has been presented almost entirely from the theoretical standpoint. It would have carried much more weight with blast-furnace men had he presented a few examples based upon accurate data of furnaces, operating along lines he claims to be essential for maximum fuel economy, namely, "Gruner's Ideal." Had such furnaces been doing consistently better work than those shown in Table I, much force would be added to his argument.

VALUE OF DIRECT REDUCTION

All the furnaces in Table I are doing direct reducing. All those that are burning less than 1,350 lb. of carbon at the tuyères are not making enough CO to reduce all the Fe_2O_3 and hence, some Fe_2O_3 must be reduced directly by carbon, and, as Prof. Richards points out, this is done three times as efficiently from the standpoint of carbon required, as when done by the indirect method. Certainly as we approach the end of the list, we find that our direct reduction is absolutely essential for there are furnaces toward the end which are not gasifying enough carbon, all told, to reduce the ore, had this carbon first been burned to CO.

I believe that Prof. Richards is absolutely right in his statement on p. 253 of *Metallurgical Calculations* when he says:

"The ordinary furnace produces at the tuyères, in order to get heat enough to melt down the charges, more CO gas than is needed to abstract all the oxygen from the charges; under these conditions it is uneconomical to oxidize any carbon at all above the tuyères. The exceptional furnace, because of pure ores, small amount of slag, pure fuel, high temperature of blast, or dry blast, gives heat enough at the tuyères

³ Thermal Principles of the Blast Furnace, *Metallurgical and Chemical Engineering*, vol. xiii, No. 16 (Dec. 15, 1915).

to melt down the charges without producing enough CO gas to reduce all the charges; under these conditions, more or less reduction is effected by solid carbon and with the greatest economy in quantity of carbon required in the furnace."

I am sure that the best furnaces, of which I have knowledge, persistently violate Gruner's ideal. Furthermore, the way we are getting our lowest coke records today is by doing a great deal of direct reduction.

Even if we were to admit that the "corrupted" Gruner's theory is correct, it is of little use unless the furnace is acquainted with the theory and willing to work along this theoretical line.

We might illustrate the case by relating the old story of the boy, the dog and the man—the furnace being in this case the dog:

The dog was barking loudly at the boy and giving every evidence of being about to attack him. The man remarked to the boy, "Oh, go ahead, he won't bite you!" The boy said, "I know it;" the man, "Well why don't you go ahead;" the boy, "Well, you see I know it and you know it, but the question is—does the *dog* know it?"

CONCLUSION

This article has developed along lines which will, undoubtedly, be criticised by some as being of but little practical value. We are, however, able to draw conclusions which will eliminate some of the factors in the question—why does one coke do better work than another? The conclusions would be that it is *not* because:

First: There is any difference in the percentage gasified at the tuyères;
or

Second: There is any difference of wind required per pound of coke;
or

Third: There is any great difference in the carbon content of the coke.

Having eliminated these three points from consideration, we must look elsewhere in endeavoring to answer our question.

As noted before, all of our lowest coke furnaces are doing some direct reduction. The heat to enable this direct reduction to be carried on efficiently must come from the carbon burned to CO at the tuyères.

It seems clear, therefore, that in low-coke furnaces, one of the most important, if not the most important function of the carbon burned at the tuyères, is to produce heat to enable the carrying on of the direct reduction, rather than to produce CO for indirect reduction.

On this basis, it becomes very essential that our carbon shall burn instantaneously to CO in order that the resulting heat may be localized where needed. This should not be a question of seconds but of a fraction of a second. If our carbon is of such a nature that this burning to CO is a comparatively long process, more of it will be required than of the quick

burning carbon in order to obtain the same concentration of heat at the desired point.

We would, therefore, say that the most desirable thing about a coke is that quality in the carbon which will allow of its being instantaneously burned to CO and thus result in the maximum concentration of heat where needed.

Practically all of the furnaces in Table I using less than 1,700 lb. of coke were using a coke made either wholly or largely from low ash, high volatile coal. While the carbon content is high, it evidently is not the quantity but the quality of the carbon that produces the low-coke consumption.

The low-coke consumption goes hand in hand with low wind. The lower volume of gas resulting means, in a hearth of a given area, a lower velocity of the gases, thus tending toward concentration of heat.

Following the same line of reasoning, we see the value of our big hearth areas and steep bosh angles, both of which tend to decrease the velocity of the gases and concentrate the heat where we need it.

The low wind and large hearths are comparatively easy of obtainment. The great problem will be to operate our coke ovens in such a manner as to obtain coke from inferior coal, the quality of whose carbon shall approach that of the carbon in the coke made from better coals. Judging from the advancement made in byproduct oven operation during the last 10 years, we have every reason to anticipate the correct solution of this problem.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Liberty Bell Methods of Precipitate Refining

BY A. J. WEINIG,* E. MET., TELLURIDE, COL.

(Arizona Meeting, September, 1916)

THE Liberty Bell cyanide precipitate is unique in that it is apt to vary widely in composition in the course of very short periods of time, and a method of refining and melting that would prove highly satisfactory at one cleanup would yield almost an hopeless "mess" at a second. Table I gives a general conception of the wide range in grade and composition of those raw precipitates. It is evident that no one standard flux could be expected to produce a high-grade bullion with such a widely variable precipitate, and that any method of treatment with acid requires most careful consideration.

Refining by Acids

Naturally, our first efforts to improve our precipitates were directed along the lines of acid treatment. Table II embodies the results of careful sampling and analysis of one cleanup at the time when wet refining reached its maximum development at this mill. We have now discarded all methods of wet refining, successfully replacing them by a more scientific method of flux calculation described herein, but it is believed that a general description of this wet refining is warranted as it throws some interesting side lights on acid refining in general.

TABLE I.—*Range of Composition of Raw Precipitates*

	Per Cent.
Gold and silver.....	25.0 to 75
Zinc.....	18.0 to 30
Lead.....	0.5 to 52
Copper.....	0.5 to 20
Silica.....	1.0 to 5
Calcium oxide.....	4.0 to 8
Sulphur.....	0.5 to 8

Column A gives the analysis of the raw precipitate; column B, that of the precipitate after heating with steam, with frequent addition of hydrochloric acid until a point was reached where the batch had a strong tend-

* Mill Superintendent, Liberty Bell Gold Mining Co.

ency to remain acid. The bath was then diluted with hot water, and repeatedly washed by decantation, until the liquor by test indicated that all soluble lime salts were removed. After the last decantation, commercial sulphuric acid was added, the bath heated by steam, and agitated with air until all action ceased. The bath was then diluted with hot water, and repeatedly washed by decantation, until by test all soluble zinc salts had been removed.

TABLE II.—*Composition of Products at Different Stages*

Substance	Raw Precipitate, Per Cent.	After Partial Treatment with Hydrochloric Acid, Per Cent.	After Hydrochloric and Sulphuric Acid Treatment, Per Cent.	After Treatment with Hydrochloric and Sulphuric Acids Followed by Caustic Soda, Per Cent.
Au and Ag	40.5	50.1	49.9	69.4
Zn	20.3	12.5	2.4	3.2
Pb	20.1	25.0	24.9	8.5
Cu	2.0	2.4	2.3	3.2
SiO ₂	1.1	1.4	1.4	1.7
CaO	4.9	0.2	Trace	
Total sulphur	1.81	1.84	5.53 ¹	3.03 ²
Total	90.71	93.44	86.43	89.03
Final weight of precipitate derived from 100 lb. of raw precipitate, pounds.	100	80	81	59
	¹ 3.51 per cent. of this sulphur was combined as sulphate.			
	² 2.55 per cent. of this sulphur was combined as sulphate.			
Column	A	B	C	D

Column C gives the analysis of the precipitate after the sulphuric acid treatment. This analysis is unique. We note first a slight decrease in the grade of the precipitate in spite of the fact that a large portion of the zinc has been removed. The increase in total sulphur content is indicative, as it shows how well finely divided lead reacts with sulphuric acid to form lead sulphate. In this case a large proportion of lead sulphate was formed, thereby actually lowering the grade of the precipitate in spite of the removal of zinc. This explains why it often happens in treating with sulphuric acid raw precipitate containing much lime, as well as lead, that a precipitate is obtained actually lower in grade than the original raw precipitate.

Column D shows analysis of the precipitate after being treated by hydrochloric and sulphuric acids and lastly with hot caustic soda. To the residue, after decanting the last sulphuric acid wash, sufficient caustic soda was added to make a strong solution. The batch was then vigorously heated and agitated with steam for 2 hr., diluted with hot

water, and repeatedly washed by decantation, until, by test, all soluble lead salts had been removed.

The effect of the caustic soda bath is well shown by the final analysis. The grade was well brought up and a marked decrease in the quantity of lead and sulphur shows the solubility of lead sulphate in hot caustic soda.

Refining by Fusion

The present method of flux refining has been arrived at through a vast amount of experiment. In general, it embodies the principle of the addition of an oxidizer, and the proper proportionment of fluxes to carry off the oxidized impurities as metallic bases in a fluid slag.

A rather wide experience in the melting of precipitates has shown that the various impurities of precipitates are oxidized and driven into the slag in a definite order. Zinc tends to oxidize first, and the resulting zinc oxide will combine with the acid radical of the flux and therefore enter the slag. As in blast-furnace practice, the zinc tends to render slags infusible; and for this reason borax glass is always used in a greater amount than silica to furnish the acid radical for the slag. Borax glass greatly increases the fusibility of zinky slags.

Sulphur tends to oxidize second, and if an alkaline oxide is present in the melt, alkaline sulphate is formed, which is rather easily fusible. This sulphate is not soluble in the slag, and, being lighter, rises to the surface of the melt, forming the well-known sulphate "cover." The sulphate "cover" is nearly always quite pure, so that after any melt the melter may easily separate it from the slag, and, by weighing and computing its sulphur content, obtain a close check on the analysis of the precipitate. This cover is either Na_2SO_4 or K_2SO_4 depending on the alkaline oxide present. (The sulphate cover is an intensely interesting material to the chemist. Many unsuspected elements are known to concentrate therein. In the Liberty Bell material, Mo, As, Sb, Te, Se, and V have been detected.) Where precipitate is melted without oxidizers, sulphur generally makes itself known by the formation of matte—a very undesirable byproduct.

After all sulphur is oxidized, lead will oxidize, and the resulting lead oxide enters the slag proper. It is well known that lead oxide combines with silica in nearly all proportions under the influence of heat, and that the resulting lead silicates are easily fusible between rather wide limits. For this reason more silica may be used in the melt when much lead is present, and for the same reason the amount of borax glass may be diminished.

After lead, copper will oxidize to Cu_2O and combine with the acid radical of the slag. The color of such slags is red. Where an excessive

amount of oxidizer is present, copper will oxidize to CuO , and slag containing this oxide is green.

This selective oxidation is most clearly evident in the order of oxidation between copper and lead. It is practically impossible to drive copper into the slag as long as metallic lead is present in the bullion. Here quantitative separations between lead and copper are easily obtained.

Graphite crucibles have a marked reducing action on lead slags. No matter how rapid the melting, or how fast the resulting slag is removed from the crucible, if much lead is present, some will be reduced to metal. In such cases it is practically impossible, in direct melting, to slag copper, even though oxidizers are added far beyond the calculated requirement.

Where melts are conducted in clay or clay-lined graphite crucibles, this reducing action is of course avoided, and as a result copper will be found to enter the slag as easily as lead, though not until all lead is oxidized. Hence, in melting precipitate in graphite crucibles no attempt is made to slag copper. If a desirable grade of bullion can not be made unless copper is slagged, the melter will then, of course, use clay or clay-lined crucibles.

As an illustration of this, I give the data regarding a melt of precipitate containing a rather high amount of copper. This precipitate had received hydrochloric acid treatment only, and gave upon analysis the following composition:

	Per Cent.
Gold and silver	68.3
Zinc.....	8.7
Lead.....	8.3
Copper.....	9.5
Sulphur.....	1.1
Calcium oxide.....	0.1
Silica.....	0.9
Total.....	96.9

All of this precipitate having been fluxed alike with an oxidizing flux equal portions were melted down in graphite and clay-lined crucibles. The resulting bullion from the clay-lined crucible shows 982 total fineness, while the bullion from the graphite crucible showed 812. I should add that, up to date, we have not found an entirely satisfactory clay or clay-lined crucible. At best they last one heat.

Fluxes.—The common oxidizing fluxes found on the market are manganese dioxide, potassium and sodium nitrate. Manganese dioxide can be obtained quite pure but at best it is costly because of its comparatively low quantity of available oxygen. It also requires extra weight of acid flux to retain the NaO in the slag. However, it gives

satisfactory results. Potassium nitrate is well known and needs but passing notice. Sodium nitrate is highly recommended and deserves special mention. It is very high in available oxygen and the crude product from Chile can be obtained practically pure so far as fluxing is concerned. It is layed down at Telluride at less than one-fifth the cost of potassium nitrate, and is one of the very cheapest fluxes on the market. It is also metallurgically more to be desired in the fluxing of sulphur, the resulting sodium sulphate being decidedly more fusible than the corresponding potassium salt and as a result the separations are better. The reaction between sodium nitrate and sulphur is $2\text{NaNO}_3 + \text{S} = \text{Na}_2\text{SO}_4 + 2\text{NO}$. Sodium carbonate or soda-ash is more desirable than the bicarbonate, since it goes considerably farther weight for weight as a flux, and lowers the freight bill. We use this flux only to supply alkali for the sulphur in the melt, when manganese dioxide is the oxidizer. Silica is an excellent acid flux, though its use must be guarded. Slags high in silica are not well adapted to the melting of precipitate, because of the higher temperature they require, especially in the presence of zinc. By reason of its cheapness, it is desirable to use as much silica as possible to replace borax in the melt, but our experience leads us never to go below a ratio of borax glass to silica in the slag of 2 to 1, where much zinc is present, or below a ratio of 1 to 1, even, where much lead is to be slagged.

In Table III, I have embodied results consistent with our practice and have arranged the same for convenience and rapidity of flux calculation.

Explanation of Table III.—In the substance column, I have placed the common impurities of the precipitate as well as the more frequently used fluxes. Column 1 contains the molecular weights of the substances in round numbers. Columns 2, 3 and 4, I have labelled "Per Cent. Oxygen." These columns indicate the weight of oxygen that the substance will combine with or evolve in per cent. of the original weight of substance. The terms "acid," and "base" oxygen I use for brevity. This is an effort to distinguish between oxygen that will enter the acid or basic part of the slag. The metallurgist's conception of a slag is that it is a fusible mixture of definite chemical combination between various acid and basic oxides. By "available oxygen" is meant the quantity of oxygen which the substance will evolve and which will be available in oxidizing metals or other impurities in the precipitate.

Column 5 gives the weight of manganese dioxide required to oxidize a unit weight of the substance on the same line in Column 1. These figures are derived from the chemical equation:

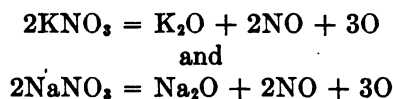


With pure manganese dioxide these figures represent approximately the amount of oxidation obtained in practice.

TABLE III.—*Flux Table*

Substance	Molecular Weight	Per Cent. Oxygen			1 Lb. Substance Requires, Pounds				
		Acid	Base	Available	MnO ₂	KNO ₃	NaNO ₂	K ₂ CO ₃	Na ₂ CO ₃
Cu''.....	63		25.4	12.7	1.40	1.07	0.91		
Cu'.....	63		12.7		0.70	0.54	0.46		
Pb.....	207		7.7		0.43	0.33	0.28		
Zn.....	65		24.6		1.35	1.03	0.87		
Fe.....	56		28.5		1.57	1.20	1.01		
S.....	32				8.30	6.30	5.30	4.30	3.30
SiO ₂	60	53.4							
CaO.....	56		28.5						
MnO ₂	88		18.2	18.2					
NaNO ₂	85		9.4	28.2					
KNO ₃	101		7.8	23.8					
Na ₂ CO ₃	106		15.1						
K ₂ CO ₃	128		12.5						
Na ₂ O.....	62		25.8						
K ₂ O.....	94		17.0						
Na ₂ B ₄ O ₇	202	47.7	7.9						
Mixture of 2 parts borax glass and 1 part silica.....		49.6	5.3						39.0 per cent. available acid oxygen.
Mixture of 1 part borax glass and 1 part silica.....		51.1	3.2						44.7 per cent. available acid oxygen.
Mixture of 3 parts borax glass and 2 parts silica.....		50.0	4.8						40.4 per cent. available acid oxygen.
Column No.....	1	2	3	4	5	6	7	8	9

Columns 6 and 7 are similar to Column 5; they represent the weights of KNO₃ and NaNO₂ required to oxidize a unit weight of substance. The figures are derived from the equations:



Authorities seem to agree that these equations represent the breaking up of the niters in presence of an oxidizable substance with heat; and I find that this agrees well with practice, where the melts are conducted in graphite crucibles. It has been frequently noted, however, that, when the melts are conducted in clay or clay-lined crucibles, the oxidizing power of the niters is markedly higher than is indicated by these equations; the reason being that graphite has a reducing action on slags. When clay or clay-lined crucibles are used, about 80 per cent. of the niter called for by the above equations will give the same oxidation in practice.

Columns 8 and 9 give the weight in pounds of potassium carbonate or sodium carbonate required to form potassium sulphate or sodium

sulphate with 1 lb. of sulphur which has been oxidized to SO_3 . Thus $\text{K}_2\text{CO}_3 + \text{SO}_3 = \text{K}_2\text{SO}_4 + \text{CO}_2$. We use these carbonates only when manganese dioxide is the sole oxidizer.

Column 10 indicates the quantity of available "acid" oxygen in mixtures of borax and silica. This table could be carried out infinitely but I have only listed mixtures that experience and practice indicate to be feasible.

In "Flux Calculation, Case I," given in the appendix, I have gone through the regular flux calculation with some detail. For the type precipitate I have taken the lowest grade of raw precipitate that we have yet melted.

The analysis of this precipitate was:

	Per Cent.
Au and Ag.....	25.0
Pb.....	38.8
Zn.....	20.5
Cu.....	1.4
S.....	1.2
CaO.....	2.0
SiO ₂	1.4
Total.....	90.3

It will be noticed that I use a type slag of 2 of acid oxygen to 1 of basic oxygen. This is the result of experience. Such slags have just a noticeable tendency to string and are the best, most easily fusible, and least corrosive to the crucible. They are invariably of low grade, and have been made repeatedly as low as \$20 per ton in precious metal. This type of slag has been found by far the most desirable, and we now make it as a regular practice, irrespective of the composition of the precipitate or fluxes used.

Since 100 lb. of precipitate was assumed at the start, 92 per cent. of this flux was added to the estimated dry weight of precipitate. It is interesting to note that this melt gave a resulting bullion fineness of 972 total fine. As we assumed that all the copper would go into the bullion, and that the balance of the impurities and base metals were removed in the slag we did expect a bullion of $25^\circ + (25^\circ \div 1.4) \times 1,000 = 950$ total fine, approximately. This melt, of course, was conducted in a graphite crucible.

"Flux Calculation, Case No. 2" was the calculation for the precipitate treated with caustic soda, listed in Column D Table II. The bullion made was 942 total fine, which is in remarkable accord with the calculated fineness, assuming that all copper went into the bullion, as $69.4 \div (69.4 + 3.2) \times 1,000 = 955$ total fine.

The calculation shows the use of soda ash and manganese dioxide.

It is also interesting that a distinction was made between sulphide and sulphate sulphur.

Sintering.—After removing the precipitate from the press the computed flux is added and the whole well mixed while wet. The mixture is then loaded into cast-iron pans, and placed in a large cast-iron muffle furnace; and the temperature of the furnace is gradually raised to a red heat. In this operation moisture is removed, and the precipitate gradually agglomerates and settles to about one-third of its original volume. During this process practically all the chemical reactions are completed and the mass sinters to a point where subsequent "dusting" is avoided.

When sintered the charge is taken from the furnace and cooled, and is easily removed from the pans. This gives an excellent product for crucible melting. In one case, where a precipitate containing 90 per cent. of gold and silver was obtained by acid-treatment, the melting of over 11,000 oz. of fine bullion was done in 4 hr. in one No. 275 oil-burning tilting-furnace. This included the time taken in starting up the furnace in the cold. With precipitate of average grade, *i.e.*, 65 per cent. Au and Ag, a melting rate of 1,300 oz. of bullion per hour is common.

The advantages gained by sintering are many. The completion of practically all chemical reactions in the muffle furnace prevents any excessive boiling in the crucible, and the crucible may be loaded to the top during the period of melting. The slow application of heat in the muffle furnace gives ample chance for complete chemical reaction. Hence, we arrive, in the use of oxidizers, at practical results which correspond with theory. The great decrease in the volume of the precipitate during sintering enables us to get much more weight in the crucible at a charge, and this again decidedly increases the melting rate. Metal losses during the sintering are exceedingly small. At one time the vents of our muffles were connected to a dust chamber, but nothing was ever recovered; and frequent tests of the gases leaving the muffles, by passing them through water or cotton soaked in a solution of sodium sulphide, failed to show more than traces of valuable metal.

These flux calculations have now been used for a period of 18 months, during which time about 60 melts have been conducted on all grades of raw and acid-treated precipitate; and in no case has there been a failure to realize a close approximation to the grade of bullion calculated.

Slags.—The slags vary so widely in composition that at a glance no relationship seems to exist. They all, however, conform to one single relationship: namely, the ratio of "acid" to "base" oxygen is approximately 2:1.

After reviewing the analysis of some 20 slags, I find that the range of the composition of slags may be considered to fall within the following

limits. The slags were made from all grades of raw and acid-treated precipitate.

	Per Cent.
Zn.....	5.0 to 30
Pb.....	0.5 to 65
Cu.....	0.0 to 20
CaO.....	0.0 to 10
SiO ₂	8.0 to 20
Mn.....	0.0 to 25
B ₂ O ₃	10.0 to 20
*Na ₂ O.....	0.0 to 18
K ₂ O.....	0.0 to 20

The composition of the sulphate cover remains quite constant.

When soda niter or sodium carbonate are used the sulphur in the sulphate cover is from 20.5 to 21.5 per cent. This is close to the theoretical requirement of sulphur in Na₂SO₄. When potassium niter is used the sulphur content lies between 17 and 18 per cent., likewise close to the theoretical requirement for sulphur in K₂SO₄.

As an illustration of the distribution of the bases and compounds in all the products of the melt, I will take Case 1. Here 134 lb. of slag and 5½ lb. of cover were made by actual weights for every 100 lb. of dry precipitate melted.

A partial analysis of this slag gave:

	Per Cent.
Zn.....	15.4
Pb.....	28.0
Cu.....	0.2
CaO.....	1.7
SiO ₂	14.1

The sulphur in the cover was not determined. If we therefore compute the actual weights of materials in precipitate and flux and in the slag, we arrive at the following figures:

Precipitate and Flux 100 Lb. Precipitate	Pounds	Slag 134 Lb.	Pounds
Zn $0.2050 \times 100 =$	20.5	Zn $0.154 \times 134 =$	20.60
Pb $0.3880 \times 100 =$	38.8	Pb $0.280 \times 134 =$	37.50
Cu $0.0140 \times 100 =$	1.4	Cu $0.002 \times 134 =$	0.27
CaO $0.0200 \times 100 =$	2.0	CaO $0.017 \times 134 =$	2.30
SiO ₂ $0.0140 \times 100 =$	1.4	SiO ₂ $0.141 \times 134 =$	19.00
In flux	18.0		
SiO ₂ Total	19.4		

If we assume that the sulphate cover contains an average of 21 per cent. sulphur, the 5.5 lb. of cover would indicate $5.5 \times 0.21 = 1.16$ lb. sulphur or 1.16 per cent. of dry precipitate. This is a very close check to the 1.2 per cent. obtained by actual analysis; and we thus arrive at

* Estimated from weights of borax and alkalis used.

results that are entirely satisfactory within the limits of experimental error.

Slag Balance			Oxygen	
Substance	Weight, Pounds	Combined as	Acid, Pounds	Base, Pounds
Zn	20.60	ZnO		5.10
Pb	37.50	PbO		2.90
Cu	0.27	Cu ₂ O		0.03
CaO	2.30	CaO		0.66
SiO ₂	19.00	SiO ₂	10.2	
*B ₂ O ₃	27.00	B ₂ O ₃	18.5	
*Na ₂ O	22.50	Na ₂ O		5.80
			28.7	14.49

* Computed from weights of borax and niter used.

Totals in the original flux calculation as 28.89 : 14.50 or a ratio of approximately 2 to 1.

The cyanide slags are stored, and once a year are melted up in a small blast furnace with brick charges, made from the ash obtained by burning old mill launders and tanks. This furnace which has a capacity of 5 tons of slag and brick a day also handles copper scrap directly, and will deliver about 15 tons of pig copper per day. This local smelting lasts only about 10 days a year. Any matte made is controlled by running the furnace more oxidizingly and thus driving the metals into the bullion and slag. As a result of this the only byproducts shipped are lead bullion and pig copper.

In spite of the seemingly high costs of local smelting, the ultimate gain is greater, by reason of the invariably greater recovery, or compared with that realized by shipping the material away.

APPENDIX

Flux Calculation Case No. 1

Data and Explanatory	Pounds				
	Oxygen		NaNO ₃	Borax Glass	Silica
	Acid	Base			
(References apply to Table III) assume 100 lb. of precipitate					
Zinc 20.5 lb.					
20.5 × 0.87 (See Col. 7, opposite Zn) =			17.90		
20.5 × 0.246 (See col. 3, opposite Zn) =		5.04			
Lead 38.8 lb.					
38.8 × 0.28 (See Col. 7, opposite Pb) =			10.90		
38.8 × 0.077 (See Col. 3, opposite Pb) =		3.00			
CaO 2.0 lb.					
2.0 × 0.285 (See Col. 3 opposite CaO) =		0.57			
SiO ₂ 1.4 lb.					
1.4 × 0.534 (See Col. 2, opposite SiO ₂) =	0.75				
Borax Glass (as it is desirable to maintain a ratio of borax to silica of 2 to 1 in the slag it is convenient to add here the proportionate amount of borax to balance silica in the precipitate. In this case the silica is low and this correction could readily be omitted on this particular precipitate but as it is necessary to correct for borax here when silica is high the correction for silica is carried through for form). Hence 2 × 1.4 = 2.80 lb. borax					
2.80 × 0.079 (See Col. 3, opposite Na ₂ B ₄ O ₇) =		0.22		2.80	
2.80 × 0.477 (See Col. 2, opposite Na ₂ B ₄ O ₇) =	1.34				
The total pounds of NaNO ₃ added is			(28.80)		
As this places extra basic oxygen in the slag as Na ₂ O we have:					
28.80 × 0.094 (See Col. 3, opposite NaNO ₃) =			2.71		
Total acid and basic oxygen		(2.09)	(11.54)		
The 2.09 lb. acid oxygen will balance $\frac{1}{2}$ (2.09) = 1.05 lb. base oxygen. This leaves 11.54 - 1.05 = 10.50 lb. base oxygen to be satisfied, or 21.0 lb. extra acid oxygen to be added. Using the mixture of 2 borax and 1 silica to supply this acid oxygen, we have 21 ÷ 0.39 (See Col. 10, opposite Mix. 2 B.G. to 1 SiO ₂) or 54 lb. of mixture. This is:					
36 lb. borax glass				36.0	
18 lb. silica					18.
also					
54 × 0.496 (See Col. 2, opposite Mix. 2 B.G. to 1 SiO ₂) =	26.80				
54 × 0.053 (See Col. 3, opposite Mix. 2 B.G. to 1 SiO ₂) =		2.86			
Grand totals of acid and basic oxygen or a ratio of 2 to 1 approx.		28.89	14.50		
Sulphur—As all sulphur goes into the "cover" as well as the niter used to oxidise it, we have: 1.2 × 5.30 (See Col. 7, opposite) =			6.37		
Grand totals of fluxes to be used			35.17	38.80	18.0

The melter made up the flux by parts as:

	Parts
NaNO ₃	35
Borax.....	39
Silica.....	18
Total.....	92

Flux Calculation Case No. 2

Data and Explanatory	Pounds					
	Oxygen		Na ₂ CO ₃	MnO ₂	Borax Glass	SiO ₂
	Acid	Base				
(References apply to Table III) assume 100 lb. of precipitate						
Zinc 3.2 lb. 3.2 × 0.87 (See Col. 7, opposite Zn) =				2.80		
Lead 3.2 × 0.246 (See Col. 3, opposite Zn) =		0.79				
8.5 lb. 8.5 × 0.43 (See Col. 5, opposite Pb) =				3.65		
8.5 × 0.077 (See Col. 3, opposite Pb) =		0.65				
Silica 1.7 lb. 1.7 × 0.534 (See Col. 2, opposite SiO ₂) =	0.89					
Borax 2.55 lb. As the slag will be low in zinc we choose a higher ratio in the slag of borax to silica for economy. Taking the ratio of 3 of borax to 2 of silica we must here take into account the necessary borax to balance the silica in the precipitate. Hence: 1.7 × $\frac{1}{2}$ × 3 = 2.55 lb. borax.					2.55	
2.55 × 0.079 (See Col. 3, opposite Na ₂ B ₄ O ₇) =		0.21				
2.55 × 0.477 (See Col. 2, opposite Na ₂ B ₄ O ₇) =	1.22					
Manganese dioxide to oxidise sulphide sulphur 3.03 - 2.55 = 0.48 lb. sulphide sulphur						
0.48 × 3.30 (See Col. 5, opposite S) =				4.00		
Hence we have 10.40 lb. MnO ₂ required to oxidise metals and sulphur.				(10.45)		
Therefore the base oxygen which the MnO ₂ carries into the slag is 10.4 × 0.182 (See Col. 3, opposite MnO ₂) =		1.90				
Totals of acid and base oxygen.	(2.11)	(3.55)				
2.11 lb. acid oxygen will satisfy $\frac{1}{2}$ (2.11) or 1.06 lb. base oxygen. This leaves (3.55 - 1.06) = 2.49 lb. base oxygen to be satisfied						
2 × 2.49. (See Col. 10, opposite Mix. of 3 borax to 2 silica)						
0.404 Equals 12.3 lb. of the acid mixture.						
This is 7.4 lb. borax.					7.40	
and 4.9 lb. silica.						4.90
Also 12.3 × 0.048 (See Col. 3, opposite Mix. 3 to 2) =		0.59				
12.3 × 0.500 (See Col. 2, opposite Mix. 3 to 2) =	6.15					
Grand totals of acid and base oxygen or a ratio of 2 to 1 approx.	8.26	4.14				
Soda ash to form Na ₂ SO ₄ with all the sulphur this all forms "cover" and is not considered a portion of the slag.						
3.03 × 3.30 (See Col. 9, opposite S) =			10.00			
Grand totals of fluxes to be used.			10.00	10.45	9.95	4.90

The melter therefore made up the following flux:

	Parts
Na ₂ CO ₃	10
MnO ₂	11
Borax glass	10
Silica.....	5
Total.....	36

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Mine Accounting for Small Mines*

BY JAMES E. CHAPMAN, WALLACE, IDAHO

(Arizona Meeting, September, 1916)

THE observations here presented are those, not of an expert accountant, but of one who, while he has seen considerable service in the accounting departments of large companies, has spent more time in engineering and operating.

This paper is intended to cover, in a measure, mine accounting for small mines, as distinguished from the elaborate systems, requiring many persons in the accounting department. I shall attempt to outline a system embracing the essentials of accounting, and simple enough in form to permit one or two persons to carry it on from month to month, in sufficient detail to be able to tell quickly the grade of ore, the prices received for metals, costs per ton for mining and milling, costs per foot for development, upward or downward tendencies in costs, ore settled or in transit, cash on hand, stocks of supplies on hand, efficiency of labor, etc.

As in all accounting, there are two main divisions: that of revenue received for what is sold, and that of disbursements made for what is bought, so in mine accounting we have to consider chiefly the income derived from sales of ore or concentrates, and the expenses incurred in producing the said ore or concentrates in a marketable condition.

I. Income

The revenue of a mining company from the sale of ore or concentrates comes in the form of remittances, accompanied by settlement sheets made out by the buyer of the mine product, such as a smelting company. It is the duty of the mine accountant (or the person to whom this is assigned in connection with other work) to check these settlement sheets, against both the mine weights and assays, and the schedules of prices arranged between the mine and the smelter. As a rule, quotations for silver, lead, zinc, copper and other metals enter into the settlements, and require checking, as do also bills of lading. Each step of the arithmetical calculation is also checked.

* Originally presented at the Wallace, Idaho, Meeting of the Columbia Section, Nov. 19, 1915.

The essential figures from these sheets are entered in an Ore Record, in columns headed "Wet Tons," "Moisture Per Cent," "Moisture Tons," "Dry Tons," "Assays per Ton," "Prices per Ton," "Contents of Metals" (in ounces and pounds), "Gross Value," "Smelter Freight and Treatment," "Net Value," and possibly others, the entries being segregated by lots, and classes of ore. Stocks on hand unsettled and in transit at the end of the month are added to the settlement figures, mine assays and estimates being used to arrive at an approximate value, and from these totals are deducted similar estimates of ore unsettled and in transit at the beginning of the month. The result is the month's production, and its net value is posted to the Ledger, to the credit of ore account and the debit of the smelting company, or to the debit of head office, in case the smelter makes its payments to such an office, on which the mine draws for funds, as required to pay operating expenses.

Secondary records in the form of a shipment book, files of bills of lading, assay certificates, etc., may be kept as the accountant finds most convenient, and according to the local conditions and circumstances of the property. Their purpose is primarily to facilitate the checking of the settlements and to aid in an accurate estimate of the amount, grade and value of ore awaiting settlement.

"Miscellaneous income" is a term which will apply to all other revenue, whether from the rent of houses owned by the mining company, or the profit on the company store or boarding house, or interest, or exchange, or dividends, or the sale of junk, tailings, etc. These are usually cash items, and are taken care of in the Cash and Voucher Record, which will now be considered.

The Cash and Voucher Record, as a single book, preferably loose-leaf, has met with general approval in recent years as a combination of cashbook, journal, and voucher register, and is the only record at the bookkeeper's desk from which posting is done to the ledger. It is compact, convenient, and desirable, serving the purpose of gathering numerous items into their proper sections at original entry, and requiring that they balance before they are used in further calculations. This book may have columns for cash, bank accounts, operating accounts, and various non-operating ledger accounts. It seems scarcely necessary to do more than sketch the use of this record, since its cash columns correspond exactly to those of a cashbook, and journal entries are made in it double, as in a journal, the only difference being that the amounts are listed in columns so that they are easier to refer to and total. Only totals are posted to the ledger, so that posting is a short and simple matter. This is done but once a month, after closing the month's operating accounts, and the purpose is to prove the balance between assets and liabilities.

II. *Operating Expenses*

Next to profits, the figures of greatest interest to a mine operator are doubtless the operating costs. The two, profits and costs, may almost be said to be complementary, and many mine operators do not doubt that if they hammer down their costs, their profits will go up. But it is capable of demonstration that an additional expenditure per ton may, under certain conditions, increase the grade of smelting ore or mill feed to a point where profits will be increased by more than the extra outlay.

For the measurement of efficiencies and the planning of improvements, a study of costs is necessary. The record of these, as often kept, not only has no statistical value, but may even be misleading, so that when work is based upon the data they furnish an expected saving will show up as a loss instead. This is particularly true of development work. Sinking a shaft, for instance, is an operation into which many factors enter to determine the true cost per foot of depth. Some of these factors, often not considered at all when a contract is let, are: The cost of air from compressor; the proportion of hoisting, pumping and timbering expense; labor and fuel for sharpening steel; repair parts and time spent in repairing machines.

An accounting system, to have value for the mine owner, should be arranged so that the important figures for use in estimating the cost of a certain operation, such as the above, can be taken off without undue waste of time, and also so that the upward or downward trend of the costs per ton in any or all departments can be seen at a glance. Practically, each mine needs to have its accounting method, almost as much as its plan of development, adjusted to its circumstances.

In this connection, I propose to discuss briefly two or three systems of classification of operating costs, evolved by large companies with whose methods I am more or less familiar, and to point out their adaptability in a measure to the requirements of smaller companies, whose office staffs consist of fewer persons, and whose need for detail, beyond the necessary working figures, is smaller.

In the first place, the distinction is drawn between direct and indirect operating costs, direct costs being, as the term indicates, those incurred for actual handling of ore, and indirect those necessitated by the operation of mining in general.

One of the largest operating companies has the following accounts for its smaller properties:

1. *Direct*.—(a) Development; (b) Ore Breaking and Stopping; (c) Timbering; (d) Tramming; (e) Hoisting; (f) Sorting, Weighing and Loading; (g) Draining.

2. *Indirect*.—(h) Branch Office; (i) Salaries; (j) General and New

York Expense; (*k*) Insurance; (*l*) Taxes; (*m*) Laboratory; (*n*) Marketing; (*o*) Transportation.

At larger and more complicated properties, the expenses are distributed on this outline, with certain modifications and amplifications. Ore Breaking and Stopping may have subdivisions as follows: Drill Steel; Machine Drills, Repairs and Maintenance; Mining Tools; Explosives; Lighting; and Pipe Lines and Tracks (Pipe lines for air and ventilation, and Track for tramming in stopes). Timbering may have subdivisions such as: Tools; Stulls, Lagging, etc.; and Miscellaneous. Tramming, which, as an account, includes only tramming on levels, may be subdivided as follows: Track and Fittings; Car and Motor Repairs and Maintenance; Animals and Feed; and Miscellaneous. Hoisting, which includes tramming from shaft-collar through an adit in the case of an interior shaft, may be divided into: (*a*) *Shaft*. Equipment Repairs and Maintenance; Shaft Repairs and Maintenance; and Miscellaneous. (*b*) *Adit*. Track and Fittings; Car and Motor Repairs and Maintenance; Animals and Feed; and Miscellaneous.

For a concern operating a mill, the number of accounts is increased by the addition of Sampling, Mill Experimental Work, and Milling. Only the last-named requires a further subdivision in order to give more definite information as to the work and the efficiency of the various departments and machines. A good arrangement of such subdivided accounts has been found to be: 1. Crusher Supplies; 2. Roll Supplies; 3. Huntington-Mill Supplies; 4. Hardinge- and Tube-Mill Supplies; 5. Elevator Supplies; 6. Trommel Supplies; 7. Jig, Table and Vanner Supplies; 8. Pump Supplies; 9. General Mill Labor; 10. Loading Concentrates; 11. Labor for Repairs and Maintenance.

"General Expense," if care is not exerted, may become a dumping place for items which ought properly to be distributed to accounts to which they pertain. The heading "Miscellaneous" is practically taboo, except as the designation of a department in which details are shown. Two or three large companies known to me, and probably many others, group all indirect charges under the head of "general" expenses, included in which is a subheading of "general" or "miscellaneous" expenses. For checking up the latter, a memorandum sheet is carried in the Distribution Book, on which spaces are provided for as many as 25 or 30 different items, among them office supplies, subscriptions, postage stamps, revenue stamps, janitor and cleaning, office fuel and heating, office lights and water, engineering instruments, blue-printing supplies, telephones, telegrams, traveling expenses, etc.

One well-known company keeps tally of these elusive items by providing for many of them among the indirect charges. Its list of indirect accounts contains 23 items: Salaries; Mine Foremen and Bosses; Storekeepers and Timekeepers; Watchmen; Stable; Office Supplies;

Rent; Telephone and Telegraph; Traveling; Lighting; Engineering and Surveying; Document Stamps; Employees' Living Quarters; Hospital Contribution and Expense; Fire Insurance; Employees' Liability Insurance; Legal Expenses; Taxes; New York Office; Accident Expenses; Securing Labor; Strike Expense; and Miscellaneous.

It is thus seen that there is considerable opportunity for meeting individual requirements in the naming and arranging of accounts. There is no hard and fast rule in these matters; and the preceding examples are presented merely as having been tried by experience, and having, under certain conditions, given satisfaction.

Vouchers and Entries.—The actual procedure in accounting for expense items comprises the following steps: (1) Making the voucher; (2) entry in cash-voucher record; (3) entry in distribution book; and (4) summary on cost sheet or monthly report.

The first step consists of writing the check for the net amount due on an invoice, using preferably a voucher-check form, on the reverse side of which is written sufficient detail to identify the transaction, together with the distribution of the amount of the check to various accounts.

In the second step, the total of the amount charged to the operating accounts is debited in a column with the title "Operating," and the amount of the check is credited in the proper bank column.

For the third step, a book with numbered columns is required, each number designating a certain account and applying only to that account. The individual figures making up the total operating charge on each voucher are entered, voucher by voucher, in the proper columns, each voucher occupying a line, and each line checking across to a total operating column, of which the total should agree with the total operating column in the cash-voucher record.

For the fourth step, the cost sheet summary, it has been found to be a good method to letter the totals of the operating accounts on tracings ruled in columns, each column representing a month. The account numbers and names can be lettered on the horizontal lines, and additional lines may be devoted to such items as "tons mined," "tons sorted," "tons milled," etc., and the results of dividing these tonnages into the various totals of expense, *i.e.*, the costs per ton, may be also noted. Prints from these tracings make neat monthly reports, very convenient for comparison since the figures for any item of expense, tonnage or cost per ton are shown side by side on the same line, from month to month throughout the year.

A comprehensive survey of operations is completed by making up a financial statement, by adding the totals of income (separating the amounts according to the classes of ore sold, if desired) and deducting the operating expenses therefrom. This is made up and entered as a voucher also, the balance being charged into profit and loss account.

Distribution of Power, Etc.—It may have been noticed that no reference has been made to power, or to mechanical or electrical time and supplies. This has purposely been deferred until the last, and will now be considered as illustrating the idea of redistribution. Following out this idea, all charges to these departments are divided at the end of the month according to the purpose for which the electric current or supplies were used, or the work done. For example, charges for current are pro-rated, according to kilowatt-hours or horsepower, to mine compressors, pumps, locomotives, hoist motors, framing-shed motors, sorting-plant motors, mill motors for driving various machines, etc. In a similar way, mechanics' and electricians' time may be redistributed by the use of cards specifying hours worked on this or that job, and repair parts used on the same. Thus proper charges may be made against the various motors, engines, and machines, cars, track, trolleys, lights—against any part of the plant, in fact, for the benefit of which this diverse work may be done. No hair-splitting is necessary or at all desirable. Each man can make out his own card in a few minutes at the end of each shift, and there is no reason why a high-priced shop-foreman should devote any time to doing the work of a cheaper clerk.

An adaptation of redistribution, making it much simpler and at the same time giving accurate statistics as to power costs for milling, for example, is to have a power-plant account from which mill-power costs and others, depending on local conditions, are redistributed, leaving a net amount as having been expended for mine power.

Purchases and Inventories.—Two other matters, which are closely related, naturally come under the control of the mine accountant, and call for brief reference. I refer to the ordering of supplies, and the keeping of stock and taking of inventories.

As to the first, the management often wishes to take charge of the actual placing of orders; but in many concerns the routine orders are usually handled by the accounting office, which takes care regularly of the checking of prices and the extensions on invoices. All this work is often delegated, in large companies, to a purchasing department, independent of the accounting department.

A lengthy paper might be written concerning the second subject alone. The main idea as regards stocks of supplies is to account for everything purchased, and charge it out as it is used. Two methods are in vogue, and both work well if a diligent and conscientious man is in charge of supplies; conversely, both work miserably if careless or slipshod methods are allowed.

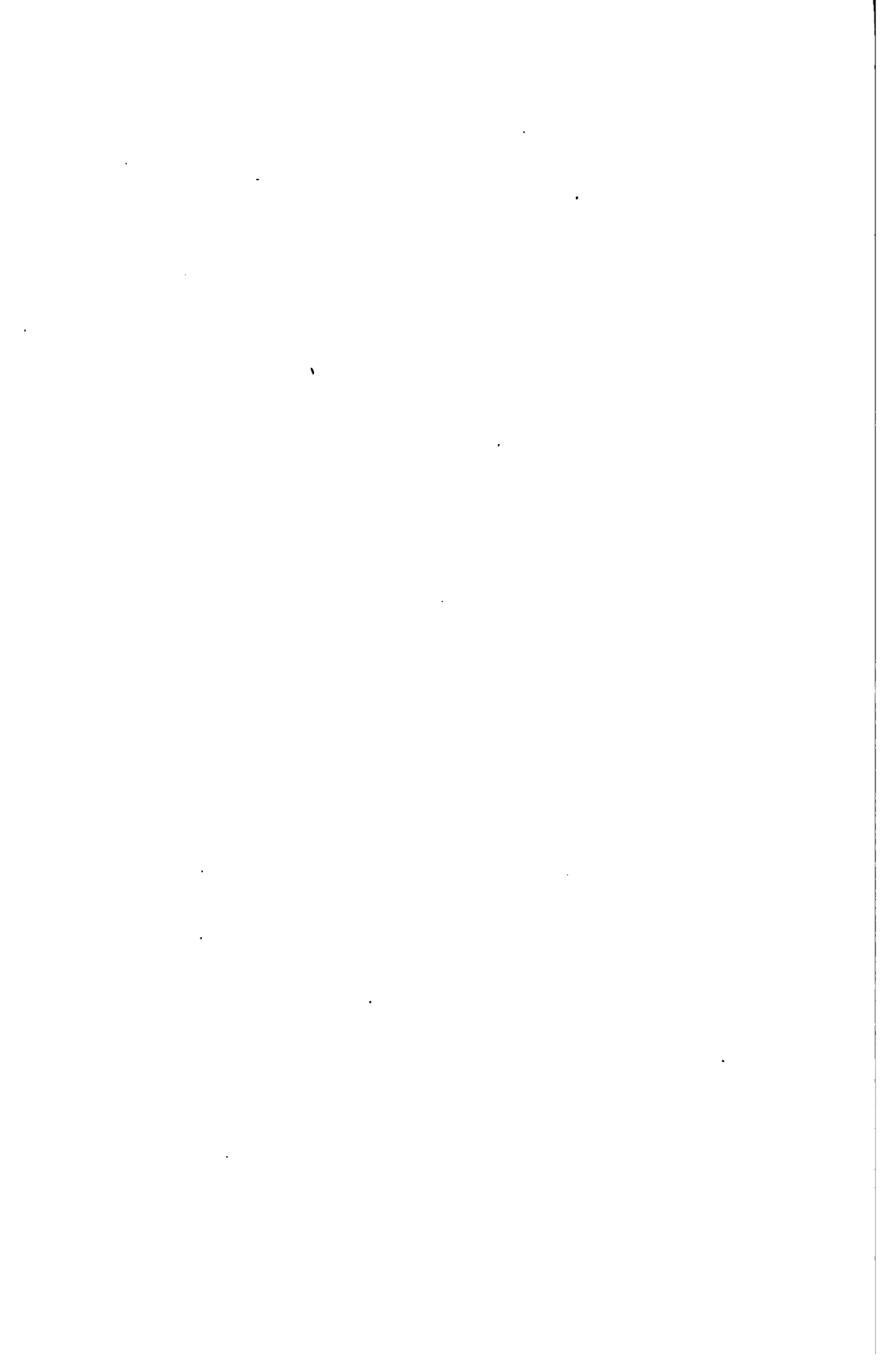
According to one method, everything is charged to stock account when purchased, and is entered in a stock ledger immediately on its delivery at the plant. When an article is issued for use, a requisition is required, specifying the account for which it is to be used, that account being

then charged and stock account being credited. The stock ledger is thus a perpetual inventory, which requires checking by actual inventory only once or twice a year.

The other method of keeping stock provides that the greater portion of purchases are charged directly to the account for the use of which they are intended, thus obviating the use of a stock ledger. Exception is made, however, in many cases, such as explosives, candles, timber, drill steel, tools, feed, hoists and parts, pumps and parts, crushers, rolls, vanners, jigs, elevators, etc., and their parts. In order to get at the true consumption in such cases, monthly inventories of these large items must be taken, the difference between stocks on hand at the first and last of a month, plus the purchases, representing the value of supplies used.

It should be borne in mind, in deciding for one or the other of these methods, that the fluctuation in costs from month to month, using the first method, is slight, because the charges for supplies in a given month are made up from the requisitions of supplies actually used out of stock during that month. Under the second method, on the other hand, the charges for supplies may vary considerably, depending as they do, directly on the invoices which are paid during the month. The first method permits paying invoices at any time without throwing the cost averages out of joint, whereas by the second method invoices must be paid just as the material is used, and furthermore the books covering a month's business must be held open until the invoices representing that month's business are received and paid.

In adopting a system of accounting, a thorough study should be made of the conditions at the mine where it is to be installed, bearing in mind also the needs and desires of the management and the stockholders in the matter of reports. Once a system has been adopted, it should be adhered to religiously, and the seeming advantages of some different system should be well considered and tried out before any change is made.



The Genesis and Relations of the Daiquiri and Firmeza Iron-Ore Deposits, Cuba

BY JOSEPH T. SINGEWALD, JR.,* PH.D., BALTIMORE, MD., AND BENJAMIN LEROY MILLER,†
PH.D., SOUTH BETHLEHEM, PA.

(New York Meeting, February, 1915.)

I. THE GENESIS OF THE DEPOSITS

THE ore deposits at Firmeza have been worked continuously since 1884; those at Daiquiri since 1895. It is surprising, therefore, that they have not been the object of careful geologic study until quite recently. It is true that considerable attention was paid to these deposits by A. C. Spencer, in 1901, in the report on the geology of Cuba, by Hayes, Vaughan, and Spencer,¹ but this was only incidental to a hasty general reconnaissance of the Island. As far as the Daiquiri deposits are concerned, this gap has been filled by the appearance of the two papers by Prof. James F. Kemp² and by Prof. Waldemar Lindgren and Clyde P. Ross respectively.³ Max Roesler, first holder of the Emmons fellowship, made a detailed study of the Firmeza deposits during the past summer, so that soon we can expect a full description of them. Consequently, there is no demand at the present time for a description of the deposits from other sources, and that is not the purpose of this paper. The papers on Daiquiri, however, leave unsolved problems in regard to the age of the rocks of the district and certain phases of the genesis of the deposits, and it is with the hope of aiding in the solution of some of these problems that this contribution is offered.

Kemp, and Lindgren and Ross agree in assigning these deposits to the type of contact metamorphic deposits. They disagree in part on the question of their localization. The deposits occur in an extensive area of diorite which includes numerous blocks of limestone, and some-

* Associate in Economic Geology, Johns Hopkins University.

† Professor of Geology, Lehigh University.

¹ C. Willard Hayes, T. Wayland Vaughan, and Arthur C. Spencer: Report on a Geological Reconnaissance of Cuba (1901).

² James F. Kemp: The Geology of the Iron-ore deposits in and near Daiquiri, Cuba. *Bulletin* No. 105, pp. 1801 to 1836 (September, 1915).

³ Waldemar Lindgren and Clyde P. Ross: The Iron Deposits of Daiquiri, Cuba. *Bulletin* No. 106, pp. 2171 to 2190 (October, 1915).

times are found in the contact zones between these blocks and the diorite, and in other instances are completely enveloped in the diorite, showing no apparent connection with the limestone. The latter class includes the larger and more important orebodies. With regard to the genesis of the former, there is no question; the difference of opinion hinges on the explanation of the latter. Kemp believes that they represent not replacements of limestone, but alterations of the diorite along lines of crushing and faulting by magmatic emissions from some underlying cooling igneous mass; whereas, Lindgren and Ross look upon them as extreme cases of the first class in which the limestone has completely succumbed to the metamorphosing influences.

Probably the first to recognize the nature of contact metamorphic deposits was B. von Cotta, in 1864, in his work on "Die Erzlagerstätten im Banat und Serbien." Deposits of the same nature were subsequently described in the Christiania region; in 1879 they were established as a special class by A. von Groddeck in his "Die Lehre von den Lagerstätten der Erze," under the name "Typus Christiania," by which they were known for a long time. But even as late as 1900, the number of examples referred to this type was not large, and, for the most part, the deposits were of little economic importance. Indeed, at that time so little importance was attached to the group that Spencer, although he considered contact metamorphism as a possible mode of origin of these deposits, readily dismissed it as not very plausible. The real birth of the type and recognition of its importance dates from the Richmond meeting of the Institute, in 1901, where three papers dealing in whole or in part with this type of deposit were presented, by J. H. L. Vogt,⁴ James F. Kemp,⁵ and Waldemar Lindgren.⁶ These papers were rightly placed in the Posepny volume on the "Genesis of Ore Deposits" published by the Institute at that time. Since their appearance, the study of contact metamorphism in its relation to ore deposition has had a popularity rivaled only by that of secondary enrichment among the subjects of greatest interest in the field of economic geology. In these investigations Kemp and Lindgren have taken a leading part, knowing this type of deposit better than any other geologists. Consequently, it is an event of unusual interest that the results of independent investigations of the same district by these two men appear almost simultaneously at this time; the fact that there is a certain divergence of opinion as to the genesis of the deposits makes a comparison of their views as expressed

⁴ J. H. L. Vogt: Problems in the Geology of Ore Deposits. *Trans.*, xxxi, pp. 125 to 169 (1902).

⁵ James F. Kemp: The Role of the Igneous Rocks in the Formation of Veins. *Idem*, pp. 169 to 198.

⁶ Waldemar Lindgren: The Character and Genesis of Certain Contact Deposits. *Idem*, pp. 226 to 244.

in the individual papers all the more interesting. Since it was our good fortune to have these papers at hand during a visit to the Firmeza and Daiquiri mines last November, it is our purpose to present here such a comparison, in which we are guided by our own observations in the field, and to state the conclusions at which we have arrived.

The interpretation given by Kemp departs from the normal one, and for that reason attention will be directed first to it. In arriving at his conclusions, he had in mind the Iron Springs deposits in Utah, and surely, though he does not mention them, those at Mackay, Idaho.

As Kemp points out, the relations at Iron Springs are the reverse of those in Cuba, since there the largest masses of ore are in the contact zone, whereas the orebodies in the andesite are not of much importance. There is, however, another great difference, in that Leith and Harder,⁷ wherever they refer to the deposits in the andesite in their report, distinctly say that they are "true veins or fissure deposits," and hence do not represent a reaction of the magmatic emissions with the constituents of the igneous rock, as do the Cuban deposits, according to Kemp's ideas. It is clear that the Iron Springs ores in andesite represent a mode of ore deposition considerably more remote from that of contact metamorphism than do the Cuban deposits.

Although a deposit of copper ore and not of iron ore, an example that is geologically more analogous is that of the Mackay district in Idaho. These deposits were described in 1907 by Kemp and Gunther,⁸ and in 1914 by J. B. Umpleby.⁹ According to Kemp and Gunther, there lies between granite and limestone a zone of quartz porphyry, the intrusion of which has produced but little effect on the limestone other than marmorization. Garnetization with associated ore deposition has taken place within the quartz porphyry in the form of pipes and chimneys. Mention is made of the occurrence of blocks of limestone in the igneous rock, but no limestone is associated with the orebodies described. These are conditions very similar to those in Cuba, viz., the presence of orebodies entirely enclosed in the igneous rock and apparently formed at the expense of that rock, and included blocks of limestone in which ore deposition did not take place. There is the additional feature of a large limestone-igneous contact along which little mineralization took place. Commenting on the peculiarity of these conditions, they say: "A new type of orebody is thus afforded." As regards the mechan-

⁷ C. K. Leith and E. C. Harder: The Iron Ores of the Iron Springs District Southern Utah. *Bulletin No. 338, U. S. Geological Survey* (1908).

⁸ J. F. Kemp and C. G. Gunther: The White Knob Copper Deposits, Mackay, Idaho. *Trans.*, xxxviii, pp. 269 to 296 (1907).

⁹ Joseph B. Umpleby: The Genesis of the Mackay Copper Deposits, Idaho. *Economic Geology*, vol. ix, No. 4, pp. 307 to 358 (1914).

ism of the process by which this garnetization took place in the igneous rock, Kemp says in another paper,¹⁰ "We are forced to the conclusion that the emissions from the deeper parts of the eruptive became charged with lime along the contact and passed upward through the igneous rock. * * * Contributions from below of iron oxide and lime have changed it to garnet and other silicates, just as the reverse contributions of silica, iron oxide and alumina to the limestone would lead to the same result."

Umpleby differs from the above geological interpretation only in regarding the quartz porphyry as a marginal phase of the granite and not a separate intrusion, and calls the whole mass granite porphyry. From his descriptions of the ore occurrences, it is seen that mining operations since 1907 have disclosed the fact that most of the apparently isolated garnet masses are in reality in contact with and part of some of the limestone blocks included in the igneous rock, but that at the same time there has been considerable endomorphic transformation of the neighboring igneous rock into garnet rock. In other words, the endomorphism has taken place in exactly the manner described in the above quotation from Kemp, but only in the immediate vicinity of the limestone inclusions. The Mackay district thus affords evidence of an endomorphic transformation of igneous rock resembling the exomorphic transformation of limestone under the action of contact metamorphism, but only at and near the contact with limestone. In the latter respect, there is an important difference between Mackay and the conditions postulated for the Cuban occurrence.

In the preceding comparison of Iron Springs, Mackay, and the Cuban deposits as interpreted by Kemp, it has been shown that, however analogous these occurrences may seem, there are important differences, and that the Cuban case is a distinct exception to the rule that might be formulated for the other two. At Iron Springs there is no endomorphism and no limestone; at Mackay there is endomorphism, but it is associated with engulfed limestone; in Cuba, Kemp postulates endomorphism in the absence of limestone. It is obvious, therefore, that the Cuban case gets no support from previous experience, but must stand on its own merits.

Endomorphism at limestone contacts has not taken place in most contact metamorphic deposits, but examples of it are sufficiently numerous and authentic not to arouse skepticism. To mention but a few additional examples, such a transformation at the limestone-andesite contact is described by Leith and Harder at Iron Springs, and at several

¹⁰ James Furman Kemp: Ore Deposits at the Contacts of Intrusive Rocks and Limestones; and Their Significance as Regards the General Formation of Veins. *Economic Geology*, vol. ii, No. 1 pp. 1 to 13 (1907).

localities in Mexico by A. Bergeat¹¹ and by J. E. Spurr, G. H. Garrey, and Clarence N. Fenner.¹²

In a recent article in *Economic Geology*,¹³ Basil Prescott reviews some facts of occurrence of contact metamorphic deposits and says, "One of the most general characteristics of the contact-metamorphic ore deposits is the development of the ores and silicates at concentrated points, the locus of which is a protrusion of the limestone into the igneous rock, or a mass of limestone included in it." That is, the engulfed blocks of limestone in the Cuban diorite afforded favorable *loci* for the most intense contact metamorphism. What is more natural to assume than that, in some cases, there has been a complete obliteration of the limestone, accompanied by considerable endomorphism, especially in view of the fact that other neighboring examples show the process in various stages of completion through the preservation of more or less limestone? The advocate of such a genesis for the Cuban deposits might deem negative arguments sufficient. The other view requires positive arguments to establish it. Let us see what the considerations are that led Kemp to his conclusions against such *a priori* reasoning.

One argument is a resemblance of polished surfaces of magnetite to a replacement of lath-shaped crystals of feldspar, suggesting the preservation of original rock texture in a replaced diorite. This is advanced merely as a resemblance, and no further emphasis laid on this point. The main emphasis is laid upon the long, narrow shape of the orebodies and their nearly vertical position, and the apparent alignment of the Lola Hill deposits and several others, lying to the south along a great belt of crushing and faulting.

It is to be regretted that both papers deal almost entirely with the Daiquiri occurrences, and that the greatest attention is paid to the Lola Hill deposits, because they are of paramount economic importance. It happens that the Firmeza mines afford exposures of great genetic significance. The Ocafia mine at Firmeza shows unmistakably the weakness of the arguments based on shape and position of the orebodies. At this mine, there are plainly visible several narrow slabs of limestone engulfed in the diorite which are oriented roughly parallel and standing

¹¹ A. Bergeat: Der Granodiorite von Concepcion del Oro im Staate Zacatecas (Mexiko) und seine kontaktbildungen. *Neues Jahrbuch*, B. B. 28, 1909, pp. 421 to 573.

¹² J. E. Spurr and G. H. Garrey: Ore Deposits of the Velardeña District, Mexico. *Economic Geology*, vol. iii, No. 1, pp. 688 to 725 (1908).

J. E. Spurr, G. H. Garrey, and Clarence N. Fenner: Study of a Contact Metamorphic Ore Deposit. The Dolores Mine at Matehuala, S. L. P., Mexico. *Idem*, vol. vii, No. 1, pp. 444 to 484 (1912).

J. E. Spurr: Theory of Ore Deposition. *Idem*, vol. vii, No. 1, pp. 485 to 492 (1912).

¹³ Basil Prescott: Some Observations on Contact Metamorphic Deposits. *Economic Geology*, vol. x, No. 1, pp. 55 to 69 (1915).

in an upright position. The ore deposit occurs in the diorite between two of these in the same position and with the same shape. One does not hesitate to consider it a replacement of one of the limestone blocks. Though not by any means as large as the ore mass on Lola Hill, this mine has furnished 300,000 tons of ore, and is estimated to contain 100,000 tons more, and is hence not an inconsiderable orebody. Other examples might be cited, as at the West No. 4 mine at Firmeza, to show that the included limestone masses tend to have the shape and position of tilted slabs.

But coming back to Lola Hill itself, where the assumption of blocks of limestone is dismissed as unreasonable in view of the extent of the orebodies, Kemp feels constrained to rely upon the presence of limestone to explain certain phenomena. On the east side of the orebody, at the south end of Lola Hill, there is a streak of garnet rock. In places this is free from ore; at other points considerable magnetite is included in the garnet, and there is more or less of a transition of the garnet rock into the ore itself through increasing quantities of magnetite. This garnet rock is explained as follows: "But in the case of the streak of garnet rock in the San Antonio with admixed calcite but no appreciable iron ore, an included slab or series of slabs of limestone is not unreasonable." Is it not equally reasonable then to extend the idea a little further, and attribute the localization of the entire orebody to such a slab or series of slabs of limestone, supplemented by more or less endomorphism?¹⁴ If we consider more closely the shape and relations of the Lola Hill occurrence, such a view appears all the more probable. It is a nearly vertical ore lens with a length of 2,650 ft., a maximum width of 250 ft., and a vertical dimension of more than 550 ft. The lens pinches rapidly in its lower part, and its termination in depth has been established in the tunnels and inclines driven in connection with the underground mining. The relations of wall rock to ore and the character of the wall rock on the bottom are the same as on the sides of the orebody. If it represents a replacement of diorite along a zone of crushing by ascending magmatic emissions, one certainly would not expect it to terminate so suddenly and at such shallow depth. On the other hand, it does look as though emissions rising through the diorite along some such zone encountered, in the place now occupied by the orebody, rock of an entirely different composition which was far more susceptible to replacement—that is, a block or blocks of limestone.

These arguments bring us to the interpretation of the genesis of the deposits that is advocated by Lindgren and Ross. The fact that in many instances smaller masses of limestone have been but partly meta-

¹⁴ Mr. Lee Reifsnider, superintendent of the mines, said that there actually is an outcrop of limestone at the south end of Lola Hill, a short distance down the slope from the San Antonio workings, which lack of time prevented us from seeing.

morphosed, whereas the much larger mass of Lola Hill has been completely replaced, they attribute to the well-known capricious character of contact metamorphism. That the replacement of limestone has been supplemented by extensive endomorphism they appreciate, and speak of the extremely energetic interchange of constituents that took place between the limestone and the igneous rock—an interchange that was so violent as to suggest "that the diorite was fluid when the metamorphism took place." Yet, one gets the impression that they fall short of attributing enough importance to the far-reaching character and particularly to the completeness of this transfer of material, for their description of the endomorphosed diorite makes it nothing more than an altered diorite and not an entirely new product, which is an integral part of the orebody, indistinguishable from the part derived from the limestone.

To sum up our opinions, the Cuban iron ores are contact-metamorphic deposits localized about engulfed blocks of limestone in diorite. In such cases, where there was a limited supply of magmatic emissions, there resulted the contact metamorphism of only a part of the limestone block. Where the supply was ample and the action most intense, not only was the block of limestone completely replaced, but complete endomorphism of the igneous rock on a large scale occurred in the vicinity. Our reasoning leads us to the conclusion that Kemp has unduly emphasized the importance of the endomorphic action and overlooked the function of the limestone in localizing that action; while, on the other hand, Lindgren and Ross have failed to emphasize sufficiently the endomorphism that played so important a rôle in the formation of many of these orebodies. We have the feeling that the true explanation lies in the middle ground between the two views.

II. THE AGE OF THE LIMESTONE

Among the many unsolved problems in the geology of the region under discussion, not the least important and interesting is that of the age of the limestone included in the diorite which encloses the orebodies. Most of those who have written on the subject of these iron ores have expressed opinions on the age of this limestone which have been little more than guesses. In 1892, H. Wedding¹⁵ referred it to the horizon of Quenstedt's *Beta* of the upper White Jura, but gave no evidence or statement of the basis upon which the correlation was made. The others have been less specific in their correlation, generally referring it to the Tertiary, and to the older Tertiary. On the geologic map

¹⁵ H. Wedding: Die Eisenerze der Insel Cuba. *Stahl und Eisen*, vol. xii, No. 12, pp. 545 to 550 (June 15, 1892).

accompanying the Index of the Stratigraphy of North America,¹⁶ only upper Cretaceous and earlier Tertiary are shown, with the exception of a coastal strip of Quaternary, in this part of the Island of Cuba. In the text of this paper (page 643) T. W. Vaughan says: "Strata have been referred to this age (Upper Cretaceous) in the literature on Oriente Province, but as no fossils have been listed there is doubt as to the extent of the area underlain by the Cretaceous in this portion of the island." No credence, therefore, is placed in Wedding's reference of the limestone to the upper Jurassic. Kemp, and Lindgren and Ross express no opinion on the subject, but Kemp says "fossils would be of extreme interest."

In view of this uncertainty in regard to the age of the limestone, and the interest attaching to its determination on account of its close association with the iron ores, we were quite surprised and highly pleased to find fossils at a small mine known as Barcelona No. 2, which lies a short distance west of the Lola Hill mines. The fossils, which include corals and sponges, were submitted to Dr. T. W. Vaughan, who kindly gave us the following information in regard to them and the age of the rock. The corals belong to the genus *Leptophyllia* (?), a species of which is found in the Cretaceous of Jamaica. Doctor Vaughan says no similar coral is known in any of the Tertiary of the United States or the West Indies. The sponges are also Mesozoic and probably Cretaceous. The age of the limestone therefore is definitely fixed as Mesozoic and probably Cretaceous.

These fossils are of importance as being apparently the first Mesozoic fossils found in this part of the Island, and more particularly as furnishing the first definite information in regard to the age of the limestone associated with the Firmeza and Daiquiri deposits. It was late in the afternoon when we visited this mine and there was little time for collecting. A more careful search than was possible in the limited time at our disposal would undoubtedly reveal other forms, as well as other localities at which the limestone is fossiliferous, and ultimately make possible a still closer correlation.

¹⁶ Bailey Willis: Index to the Stratigraphy of North America. *Professional Paper No. 71, U. S. Geological Survey* (1912).



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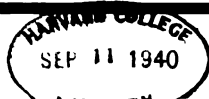
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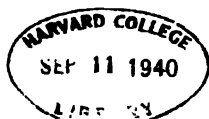
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CONSTITUTIONAL AMENDMENT RAISING DUES PASSED

The amendment to Article III, Section 1, of the Constitution, raising the dues from \$10 to \$12 a year commencing in 1917, which was submitted to the members for letter ballot, was passed by a vote of 1,039 to 401.

MANUSCRIPTS FOR THE ARIZONA MEETING OF THE INSTITUTE

The next meeting of the Institute, the 113th meeting, will be held in Arizona in the latter part of September, 1916. All papers to be presented at this meeting must be published in the September Bulletin or previously. Manuscripts must, therefore, be in the hands of the Secretary of the Institute not later than July 1. In case manuscripts are given by authors to members of technical committees, or others, they must be in the hands of such persons sufficiently in advance so that they may be actually in the hands of the Secretary of the Institute by the date mentioned.

In the past, the Secretary has received urgent requests for special consideration regarding manuscripts that come in after the closing date. In view of the obvious unfairness of this, the Committee on Papers and Publications has instructed the Secretary not to accept for presentation at the Arizona Meeting any papers received later than July 1, 1916.

PROCEEDINGS OF THE ONE HUNDRED AND TWELFTH MEETING, NEW YORK CITY, FEBRUARY, 1916

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Committee on Arrangements

DAVID H. BROWNE, *Chairman*

BRADLEY STOUGHTON, *Vice-Chairman*

LAWRENCE ADDICKS
PERCY E. BARBOUR
GEORGE D. BARRON
J. V. N. DORR
J. R. FINLAY
LOUIS D. HUNTOON

BURR A. ROBINSON
E. MALTBY SHIPP
JOSEPH STRUTHERS
E. B. STURGIS
T. T. READ
R. H. VAIL

Ladies' Committee

MRS. LOUIS D. HUNTOON, *Chairman*

MRS. LAWRENCE ADDICKS
MRS. GEORGE D. BARRON
MRS. DAVID H. BROWNE
MRS. J. V. N. DORR
MRS. ARTHUR S. DWIGHT
MRS. KARL EILERS
MRS. H. W. HARDINGE
MRS. LEVI HOLBROOK
MRS. AXEL O. IHLESEN
MRS. W. R. INGALLS
MRS. J. H. JANEWAY
MRS. SIDNEY J. JENNINGS

MRS. L. O. KELLOGG
MISS ELIZABETH H. KUNZ
MRS. A. R. LEDOUX
MRS. WILLARD S. MORSE
MRS. HENRY S. MUNROE
MRS. H. A. PROSSER
MRS. CHARLES F. RAND
MRS. THOMAS ROBINS
MRS. BURR A. ROBINSON
MRS. E. M. SHIPP
MRS. BRADLEY STOUGHTON
MRS. B. B. THAYER

Finance Committee

G. D. BARRON, *Chairman*
ROBERT M. CATLIN
W. A. CLARK
KARL EILERS
JAMES GAYLEY

C. F. KELLEY
ALBERT R. LEDOUX
W. W. MEIN
W. H. NICHOLS, JR.
WILLIAM L. SAUNDERS

Entertainment Committee

THOMAS T. READ, *Chairman*
L. D. HUNTOON

GILBERT RIGG
W. B. MCKINLAY

LAWRENCE A. ADDICKS

Registration Committee

BURR A. ROBINSON, *Chairman*
G. A. ROUSH

L. T. BUELL

College Reunion

JOSEPH STRUTHERS, *Chairman*

Banquet Committee

E. B. STURGIS, *Chairman*
N. H. EMMONS, JR.

R. C. PATTERSON, JR.

Luncheon Committee

E. M. SHIPP, *Chairman*
L. O. KELLOGG

PHILIP L. GILL

Program Committee

PERCY E. BARBOUR, *Chairman*
BURR A. ROBINSON

H. A. MEGRAW

Public Announcement Committee

R. H. VAIL

Monday, February 14, 1916

Reception Committee

L. D. HUNTOON, Chairman

W. H. ALDRIDGE	L. W. MAYER
H. P. BANKS	H. S. MUNROE
WILLIAM CAMPBELL	F. E. PIERCE
J. PARKE CHANNING	H. A. PROSSER
W. B. DEVEREUX	T. T. READ
E. L. DUFOURCQ	R. M. RAYMOND
A. S. DWIGHT	F. T. RUBIDGE
KARL EILERS	F. H. SIMONDS
A. H. ELLIOTT	J. STRUTHERS
H. W. GEPP	G. C. STONE
J. H. JANEWAY	A. L. WALKER
G. F. KUNZ	H. C. WILMOT
A. R. LEDOUX	W. Y. WESTERVELT
T. H. LEGGETT	

Tuesday, February 15, 1916

Reception Committee

J. V. N. DORR, Chairman

H. N. SPICER	J. E. JOHNSON, JR.
NOEL CUNNINGHAM	E. F. ROEBER
J. A. CHURCH, JR.	H. A. POILLON
G. D. VAN ARSDALE	J. GORDON HARDY
J. W. MERCER	W. ROWLAND COX

Wednesday, February 16, 1916

Reception Committee

J. R. FINLAY, Chairman

CLINTON H. CRANE	W. R. INGALLS
H. P. HENDERSON	SIDNEY H. BALL
C. M. WELD	ROBERT PEELE
R. A. F. PENROSE, JR.	

TECHNICAL SESSIONS

Monday Morning, Feb. 14, 1916.—The opening session, like all the technical sessions, was held at the headquarters of the Institute in the Engineering Societies' Building, New York City. It was under the auspices of the Committee on Petroleum and Gas. Dr. David T. Day, Vice-Chairman of that Committee, presided.

The following papers were presented by their authors or authors' representatives:

Development of the Law Relating to the Use of Gas Compressors in Natural Gas Production. By Samuel S. Wyer.

Necessary Use and Effect of Gas Compressors on Natural-Gas Field Operating Conditions. By Samuel S. Wyer. (Discussed by David T. Day, I. N. Knapp, W. L. Saunders, Harrison Souder, F. G. Clapp.)

The Evolution of Drilling Rigs. By R. B. Woodworth. (Discussed by Chester W. Washburne, I. N. Knapp, W. L. Saunders, Samuel S. Wyer, Leonard Waldo, David T. Day, E. Gybbon Spilsbury.)

The following papers were presented by title:

The Control of Petroleum and Natural Gas Wells. By Alfred. G Heggem. (Discussed by I. N. Knapp.)

Monday Afternoon, Feb. 14, 1916.—This session was under the auspices of the Committee on Coal and Coke, and S. A. Taylor presided.

The following papers were presented by their authors or authors' representatives:

Illumination of Mines. By Robert P. Burrows. (Discussed by Edwin M. Chance, T. M. Chance, G. S. Rice, D. B. Reger, R. V. Norris, E. T. Lednum, G. Dawson Hall, Hugh Archbold, George H. Stickney.)

Some Researches on Fire-Damp. By Enrique Hauser. (Discussed by George A. Burrell, W. E. Gibbs, E. M. Chance, T. M. Chance.)

The following papers were presented by title:

Economies in a Small Coal Mine. By H. A. Everest. (Written discussion by Newell G. Alford.)

The Effect of Aëration and "Watering Out" on the Sulphur Content of Coke. By J. R. Campbell.

Notes on Brown Coal Mining in Germany. By George J. Young.

The Stresses in the Mine Roof. By R. Dawson Hall. (Discussed by Charles Enzian, Hugh Archbold, T. M. Chance.)

Monday Afternoon, Feb. 14, 1916.—This session was under the auspices of the Committee on Geology, Professor Alfred C. Lane, presiding.

The following papers were presented by their authors or authors' representatives:

Magmatic Differentiation in Effusive Rocks. By Sidney Powers and Alfred C. Lane. (Written discussion by N. L. Bowen.)

The Iron Deposits of Daiquiri, Cuba. By Waldemar Lindgren and Clyde P. Ross. (Discussed by Joseph T. Singewald, Jr., Benjamin Leroy Miller, M. Roeder, L. C. Graton, Harrison Souder, Charles P. Berkey, Alfred C. Lane, J. D. Irving.)

Interpretation of Assay Curves for Drill Holes. By Edward H. Perry and Augustus Locke.

The following papers were presented by title:

The Disseminated Copper Ores of Bingham Canyon, Utah. By J. J. Beeson.
Geology of the Ore Deposits of the Tintic Mining District. By Guy W. Crane.
Observations on Certain Types of Chalcocite and Their Characteristic Etch Patterns. By C. F. Tolman, Jr. (The three papers named above were discussed by L. C. Graton, Alfred C. Lane, J. T. Singewald, Jr., Charles P. Berkey. Written discussion by H. E. Merwin.)

The Iron Ores of the Philippine Islands. By Wallace E. Pratt.

Tuesday Morning, Feb. 15, 1916. Annual Meeting, President W. L. Saunders presiding.

Seventy-two members were present and 1,440 voted by letter ballot.

The minutes of the Annual Meeting of Feb. 16, 1915, were read, and on motion, duly made and seconded, were approved.

Reports of the President,¹ Secretary and Treasurer² were presented in writing.

Report of the Tellers for Election of Officers and Directors was presented in writing and the following Officers and Directors were declared elected.

Director and President,
Director and Vice-President,
Director and Vice-President,

L. D. Ricketts,
Karl Eilers,
James MacNaughton,

¹ See *Bulletin* No. 111, p. vii (March, 1916).

² See *Bulletin* No. 110, pp. iv and vii (February, 1916).

Director,
 Director,
 Director,
 Director,
 Director,

George D. Barron,
 Charles W. Goodale,
 Edwin Ludlow,
 Charles F. Rand,
 Thomas B. Stearns.

The report of the Tellers on the Amendments to the Constitution was presented in writing and the amendment was declared carried by a vote of 1,039 to 401.

The reports of the following committees were presented:

Committee on Membership,
 Finance Committee,
 Library Committee,
 Committee on Papers and Publications.

A petition was presented that the question of the Institute adopting simplified spelling be submitted to letter ballot and was ordered to take the usual course.

On motion, duly made and seconded, telegrams were sent to Dr. James Douglas and Dr. James F. Kemp* expressing greetings of the members.

Tuesday Morning, Feb. 15, 1916. Technical Session, President W. L. Saunders, presiding.

The following papers were presented by their authors or authors' representatives:

Notes on Flotation. By J. M. Callow. (Discussed by R. H. Richards, J. W. Richards, Leonard Waldo. Written discussion by George D. Van Arsdale.)

Broken Hill Underground Mining Methods. By E. J. Horwood. (Discussed by Albert R. Leoux.)

The following papers were presented by title:

Grinding Brass Ashes in the Conical Ball Mill. By Arthur F. Taggart and R. W. Young.

Underground Mining Methods of Utah Copper Co. By Thomas S. Carnahan.

Tuesday Afternoon, 15, 1916.—R. M. Catlin presided.

The following papers were presented by their authors or authors' representatives:

Tests on Motor-Driven Equipment for Use in Preparation of Anthracite Coal. By H. M. Warren, A. S. Biesecker and E. J. Powell. (Discussed by R. V. Norris, William Kent. Written discussion by K. A. Pauly.)

The New Electric Hoist of the North Butte Mining Co. By Franklin Moeller. (Discussed by Graham Bright. Written discussion by K. A. Pauly.)

Conservation of Iron Ores. By C. K. Leith.

The following papers were presented by title:

An Electro-Hydraulic Shovel. By Frank H. Armstrong.

Application of Electric Power to Mining Work in the Witwatersrand Area, South Africa. By J. N. Bulkley. (Discussed by Graham Bright. Written discussion by K. A. Pauly.)

Conservation and Economic Theory. By R. T. Ely.

Pennsylvania Fire Clay. By L. C. Morganroth. (Discussed by David B. Reger.)

The Use of Low-Grade Phosphates. By James A. Barr.

Wednesday Morning, Feb. 16, 1916.—This session was under the auspices of the Committee on Precious and Base Metals, George C. Stone, presiding.

* See p. xi of this *Bulletin* for Prof. Kemp's acknowledgment.

The following papers were presented by their authors or authors' representatives:

- The Behavior of Stibnite in an Oxidizing Roast. By H. O. Hofman, and John Blatchford. (Discussed by Robert H. Richards.)
The Determination of Grain Size in Metals. By Zay Jeffries, A. H. Kline and E. B. Zimmer. (Discussed by Alfred C. Lane, C. H. Fulton, W. M. Corse.)
Substitutes for Platinum. By F. A. Fahrenwald.
The Newnam Hearth. By William E. Newnam.

The following papers were presented by title:

- Determination of Antimony in the Products Obtained by Roasting Stibnite. By W. T. Hall and John Blatchford.
Recent Advances in the Chemistry of the Cyanogen Compounds. By J. E. Clennell.
Recrystallization of Cold-Worked Alpha Brass on Annealing. By C. H. Mathewson and Arthur Phillips. (Discussed by C. H. Fulton, Zay Jeffries. Written discussion by Zay Jeffries.)
Segregation in Gold Bullion. By James H. Hance. (Discussed by Frederic P. Dewey, Francis P. Sinn, E. Gybbon Spilsbury, Smith.)

Wednesday Morning, Feb. 16, 1916.—This session was under the auspices of the Iron and Steel Committee, J. W. Richards, presiding.

The following papers were presented by their authors or authors' representatives:

- Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains. By R. J. Wysor. (Discussed by Leonard Waldo, J. W. Richards, Linn Bradley.)
The Control of Chill in Cast Iron. By C. M. Thrasher. (Discussed by Albert Sauveur, Richard Moldenke.)
Magnetic Studies of Mechanical Deformation in Certain Ferro-Magnetic Metals and Alloys. By H. Hanemann and P. D. Merica. (Discussed by J. A. Mathews, Leonard Waldo.)
Effect of Carbon on the Physical Properties of Heat-Treated Carbon Steel. By J. H. Nead. (Discussed by Albert Sauveur, J. A. Mathews, Frank N. Speller. Written discussion by E. D. Campbell.)

The following papers were presented by title:

- A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron. By W. McA. Johnson. (Discussed by J. W. Richards.)
Manganese-Steel Castings in the Mining Industry. By Walter S. McKee.
Washed Metal. By Henry D. Hibbard and Edward L. Ford.
The Iron Mines of the Sierra Menera District of Spain. By Messrs. Sota, Aznar, and Callen.

Wednesday Afternoon, Feb. 16, 1916.—This session was under the auspices of the Iron and Steel Committee, J. W. Richards, presiding.

The following papers were presented by their authors or authors' representatives:

- Modern Development in the Combustion of Blast-Furnace Gas with Special Reference to the Bradshaw Gas Burner. By K. Huessener. (Discussed by K. Nibecker, S. K. Varnes. Written discussion by R. J. Wysor, J. W. Richards.)
Vacuum Fused Iron with Special Reference to the Effect of Silicon. By T. D. Yensen. (Discussed by J. A. Mathews, W. E. Ruder, J. W. Richards.)
Metallography of Steel for U. S. Naval Ordnance. By H. E. Cook. (Discussed by Albert Sauveur, J. W. Richards, Richard Moldenke. W. E. Ruder, Leonard Waldo.)

The following papers were presented by title:

- Iron Ores of California and Possibilities of Smelting. By C. Colcock Jones.
Manufacture and Tests of Silica Coke-Oven Brick. By Kenneth Seaver. (Discussed by J. W. Richards, W. H. Blauvelt, F. G. Bryer, C. G. Atwater.)

The Duplex Process of Steel Manufacture at the Maryland Steel Works. By F. F. Lines.

The Electric Furnace in the Foundry. By William G. Krantz.

Commercial Production of Sound Homogeneous Steel Ingots and Blooms. By E. Gathmann.

Suggestions Regarding the Determination of the Properties of Steel. By A. N. Mitinsky. (Discussed by Albert Sauveur, J. W. Richards, William Kent, Leonard Waldo.)

ENTERTAINMENT OF LADIES

The complete plans which had been made for the entertainment of ladies at this meeting, and the success of the efforts of the Ladies' Committee during the past few years bore fruit in the attendance of a still greater number of ladies than in any recent year. There was also manifested a most cordial spirit of appreciation on the part of the visiting ladies, as exemplified on Thursday's boat trip by their presentation of a bunch of violets with orchid to each member of the Ladies' Committee, and also the presentation to the Chairman of the Ladies' Committee, of a large bunch of roses with an appreciative letter from one of the visiting ladies. The visiting ladies were met as usual by members of the Ladies' Committee in the Ladies' Reception Room each day from 12:00 to 12:30 o'clock and were then escorted to luncheon. The entertainment for the ladies comprised an Ice Carnival at Hotel Biltmore on Monday afternoon; a visit to the Hippodrome where, besides the usual exhibitions, there was some very fine ice skating, on Tuesday afternoon; a visit to the Art Galleries of Senator William A. Clark on Wednesday afternoon, where refreshments were served, and also tea at the residence of Mrs. Bradley Stoughton. On Wednesday evening there were 120 ladies at the Banquet and nearly this number attended the all-day Boat Excursion on Thursday.

SOCIAL FEATURES OF THE MEETING AND VISITS

Annual Dinner.—The Annual Dinner was held on Wednesday evening, Feb. 16, at Hotel Astor. The attendance included 358 members and guests and the dinner was followed by dancing. The Dance Committee was composed of:

James T. Kemp, *Chairman*,
de Courcy B. Browne,
Prof. William Campbell,
Richard B. T. Kiliani,
R. V. Norris, Jr.,
Benjamin F. Tillson,
Roger L. Winsley.

College Night.—Many colleges were represented at the meeting of the Institute and graduates who were present met together for a social evening in individual groups numbering from 6 to 60 at their College Clubs or elsewhere, including some dinners at the Engineers' Club, and especially a dinner and smoker for Columbia graduates at the Columbia University Club.

Luncheons.—On each of the three days luncheons were served in buffet fashion in the room adjoining that in which the technical sessions were held.

New York Club Privileges.—The following New York Clubs very courteously extended their facilities to visiting members during the meeting and honored cards of introduction from the Institute:

Chemists' Club	Machinery Club	Technology Club
Columbia University Club	Princeton Club	Williams Club
Harvard Club	Rocky Mountain Club	Yale Club

Smoker.—The smoker, with its unique and highly entertaining features, was a much appreciated event, regarded as an innovation by those who have attended recent meetings. It was held at the Aldine Club, 200 Fifth Avenue, and was attended by 250 men. The entertainment was entirely by members of the Institute, except for the music. The program was in charge of T. T. Read, as "mine foreman," ably assisted by James T. Kemp and other "shift bosses," undergraduates of Columbia School of Mines. It included an opening address by David H. Browne, a remarkable Talking Doll with T. B. Stearns as interlocutor; the Metallurgical Cow, explained by David H. Browne; original verses by Col. A. M. Hay; pseudo-serious lecture and demonstration of Flotation Process by Gilbert Rigg, assisted by other members, especially Dr. A. R. Ledoux, W. B. McKinlay and Burr A. Robinson; reminiscences by some of the older members, including Dr. R. W. Raymond, E. C. Pechin and Philip N. Moore; and moving pictures interspersed with songs and refreshments. A little booklet of songs was distributed to all through the courtesy of *Metallurgical & Chemical Engineering*.

Boat Excursion.—Many members and guests of the Institute assembled at the Brooklyn Navy Yard on the morning of Thursday, Feb. 17, and were taken over the Yard by guides provided by Admiral Usher. At 12:00 o'clock sharp 262 members and guests left the Navy Yard on board the S. S. "Dolphin." Luncheon was served on board and the party was landed at Sandy Hook at 2:30 p. m. More than an hour was devoted to an exhibition of the works at Sandy Hook and to watching the firing of some of the guns. The mist prevented the firing of some of the big guns on account of the danger thereof, but the visit very much interested all who took part. The "Dolphin" then brought the party back and landed it at South Ferry. Notwithstanding the season of the year, the temperature did not prove too cold to mar the pleasure of the occasion and those who were in danger of becoming chilled engaged in dancing on deck to the music of the splendid band which had been provided by the authorities. Appreciation was expressed on all sides for the courtesy of the Secretary of the Navy in extending the use of the "Dolphin" through the members of the Naval Consulting Board, who are members of the Institute, and to the Officers of the Navy and the Commandant and Officers at Sandy Hook who did so much to make the trip pleasurable and interesting.

The Daily Fume.—An account of the meeting would not be complete without saying a few words of appreciation of the Daily Fume, which was edited anonymously and issued each day by courtesy of the *Engineering and Mining Journal*. This paper was written in lighter vein and almost boasted in the first issue that accuracy was deliberately sacrificed to interest. The publication of this daily, announced as the official organ of the American Institute of Dining Engineers, did much to enliven the social features of the meeting.

NEWS FROM PROFESSOR KEMP

The following letter was received from Prof. Kemp in reply to a telegram expressing the greetings of the members attending the Annual Meeting in New York.

"Melbourne Beach, Fla.
Feb. 17, 1916.

Dear Mr. Stoughton:

The welcome telegram from the members of the Institute, which you kindly transmitted to me, came safely yesterday and I catch the next available mail with a heart-felt acknowledgment. I cannot express to you how greatly the remembrance cheers and braces me during the days of patient waiting for the feeling of over-strain to fully pass, and for the time when I can again be back among the friends in the Institute who are very dear to me. While I cannot reach all the members who joined in the telegram I will ask you to say to such as are available that I am profoundly sensible of the thoughtfulness which prompted the message.

I write from a little bungalow on the ocean side of the half mile or more of sand dunes, thickly grown with palmetto, which separate the ocean from the Indian River. Two miles or more of Indian River still intervene between the little colony of Melbourne Beach and Melbourne on the mainland and on the Florida East Coast R. R. where the telegraph station is. I was thus unable to respond at once by wire.

Sincerely yours,
(Signed) J. F. Kemp."

UNITED ENGINEERING SOCIETY

The following are excerpts from the report of the Treasurer of the United Engineering Society, for the year ending Dec. 31, 1915.

Finances

Due to the growth of the activities of the United Engineering Society, it was found desirable early in the year to consolidate the various accounts and financial records of the Society and of the Library, heretofore kept separately, and to have all the data pertaining to the building and its operation in the charge of one individual. An assistant secretary especially assigned to this work has been appointed, and a separate office has been established. The Finance Committee has had all of the records brought fully to date and has prepared a new set of account books to meet the new conditions.

The Real Estate Account now includes the following items:

Land.....	\$ 540,000.00
Building.....	1,050,000.00
Equipment.....	33,171.16
Founder Societies, Preliminary expenses.....	24,000.00
Total.....	\$1,647,171.16

The principal of the mortgage on the Land held by Andrew Carnegie amounting originally to \$540,000 has been finally satisfied in full by payments from the Land and Building Funds by the Founder Societies.

The gross operating expenses for the year 1915 were \$44,440.28, an increase of \$4,858.64 over the gross expenses of the preceding year. This increase was due chiefly to the unusual alterations on the sixth floor amounting to \$4,258.36. The total revenue was \$48,028.20.

The funds available for the uses of the Library Board during the year were \$17,444.87. The expenditures of the Library Board were \$16,415.14, leaving an unexpended balance of \$1,029.73.

During the year the Finance Committee, with the assistance of experts, has revised the assessments for space occupied by all Societies, Founders as well as Associates. In doing this, consideration has been given to location as well as area occupied, and the system of assessment made uniform. The schedule of rates is available for examination by any interested person or society.

Gifts and Endowments

On Jan. 27, 1915, Ambrose Swasey presented to the United Engineering Society the sum of \$200,000 in securities, the income only therefrom to be used for the purposes of the Engineering Foundation, for "the advance of the engineering profession and the benefit of mankind."

On Apr. 5, 1915, Dr. James Douglas presented the sum of \$5,000 to the United Engineering Society to form the nucleus of an endowment fund for the Library.

Office Occupancy

There are now fourteen Associate Societies occupying space in the building.

Meetings or Lectures

The record of the number of times the rooms were used during the year (1915) for meetings or lectures (not for office occupancy) is:

Meeting Room		Number of Times Occupied		
		1914	1915	
1st floor foyer.....		0	1	1 more
Auditorium.....	3d and 4th floors	76	42	34 less
No. 1 Assembly Room.....	5th floor	58	53	5 less
No. 2 Assembly Room.....	5th floor	94	76	18 less
No. 3 Assembly Room.....	5th floor	46	49	3 more
No. 5 Assembly Room.....	6th floor	0	2	2 more
Small committee room on.....	1st floor	18	34	16 more
Assembly Room 1201 on.....	12th floor	53	30	23 less
Total.....		345	287	58 less

Library

During the year the administration of the Library has been made the subject of an agreement executed by the three Founder Societies and the United Engineering Society under which all expenses of the Library are paid from the United Engineering Society office. Each Founder Society has contributed \$4,000 and the remainder of \$2,500 has been paid by the United Engineering Society. The total appropriation for books and administration amounted to \$14,500 for the year. In

addition there were derived from searches, \$2,410.80, and from sales and income \$534.07, making a total revenue of \$17,444.87.

In June there was effected a consolidation of the various insurance policies heretofore held separately by the Founder Societies and the United Engineering Society, into one policy covering all.

Respectfully submitted,

JOS. STRUTHERS,
Treasurer.

Assets

Real Estate.....	\$1,647,171.16
Investments—Engineering Foundation Fund.....	200,000.00
Investments—Library Endowment Fund.....	5,002.50
Investments—General Funds.....	58,138.75
Cash.....	11,307.05
Unexpired insurance.....	3,949.16
Accounts receivable.....	3,785.36
	\$1,929,353.98

Liabilities

Founders Equity in Property.....	\$1,647,171.16
Due the General Reserve Fund.....	10,000.00
Due the Depreciation and Renewal Fund.....	58,845.67
Due the Engineering Foundation Fund.....	200,000.00
Due the Library Endowment Fund.....	5,000.00
Due the Library Board—1915 unexpended balance.....	1,029.73
Surplus Dec. 31, 1915.....	7,307.42
	\$1,929,353.98

Certified by Barrow, Wade,
Guthrie & Co.

CORRECT—FINANCE COMMITTEE
CHARLES F. RAND, *Chairman*
H. H. BARNES
JESSE M. SMITH.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period Feb. 10, 1916 to Mar. 10, 1916:

D. A. S. Bell, Ottawa, Can.
John Cooper, Exeter, Devonshire, Eng.
I. Dally, Seattle, Wash.
George W. Evans, Seattle, Wash.
Alexander Fyfe, London, Eng.
F. C. Greene, Seattle, Wash.
I. Ishikawa, Tokio, Japan.
William Kelly, Vulcan, Mich.
Paul S. King, Wilmington, Del.
Oscar Lachmund, Greenwood, B. C.
J. S. Lane, New York City.

E. B. Latham, Santa Maria, Cal.
H. N. Lawrie, Portland, Ore.
L. A. Levensaler, Tacoma, Wash.
George J. Lyon, Schenectady, N. Y.
Fred MacCoy, Antofagasta, Chile.
J. Mitchell-Roberts, London, Eng.
Chas. W. A. Sultan, Yonkers, N. Y.
R. B. Tinsley, Big Stone Gap, Va.
Walter E. Trent, Reno, Nev.
Alfred W. G. Wilson, Ottawa, Can.

Nelson Dickerman has resigned as general Manager of the Pato Mines (Colombia), Ltd., and The Nechi Mines (Colombia), Ltd., and will return from Colombia to San Francisco, Cal., the latter part of April. W. A. Prichard has been appointed as his successor.

H. A. Barker, after a period as consulting engineer for the Sociedad Explotadora de Caylloma Consolidada, has accepted the management of the company's properties in Peru.

W. T. Burns, formerly general foreman of the Electrolytic Copper Refinery of the Boston & Montana, has been made superintendent of the electrolytic plants of the Anaconda Copper Mining Co., Great Falls, Mont.

James P. Cahen, Jr., has accepted a position with the Ray Consolidated Copper Co., Ray, Ariz.

Milton R. Evans has been appointed mine inspector for the Associated Companies, Plymouth, Pa.

C. S. Herzig has opened an office as consulting mining engineer at 27 William Street, New York, N. Y.

V. H. Hughes, of Valerius, McNutt & Hughes, has returned from an 8 weeks' trip in Mexico on oil and gas problems.

William F. Jahn, for the past three years Mill Superintendent for the New York & Honduras Rosario Mining Co. at San Juancito, Honduras, C. A., has resigned and will leave shortly for the United States.

E. Horton Jones has become chief engineer of the Canadian Copper Co., Copper Cliff, Ont., Canada.

Robert M. Keeney is now with the Snyder Electric Furnace Co., of Chicago, Ill.

Frederick MacCoy has resigned his position as assistant manager of the Esperanza Mining Co., El Oro, Mex., and has joined the staff of Huntington Adams, Antofagasta, Chile.

J. D. Malcolmson has been appointed a Fellow of the Mellon Institute, University of Pittsburgh, in the investigation of the problem of the recovery of copper from its low-grade ores.

The Associated Geological Engineers announce the opening of a New York office at 3112 Equitable Building, 120 Broadway, in charge of **Frederick G. Clapp**, Managing Geologist of the Petroleum Division. They will continue the practice of Geological Engineering in all its branches, with special reference to examinations and reports on oil and gas properties.

J. M. Williams, Jr., has accepted the position of superintendent of the Torreón Factory of the Continental-Mexican Rubber Co., Apartado 176, Torreón, Coah., Mex.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Junior and assistant engineers at \$100 to \$125 per month; assistant foremen and underground shift bosses at \$150 per month; wanted for a large copper company in the West. Good opportunity for advancement. No. 77.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, graduate engineer, aged 44, with over 22 years' experience in connection with the development of large coal and coke properties in different parts of the country, desires position as general manager or general superintendent of large bituminous coal producer, or will take an interest and the management of a smaller operation. No. 258.

Member, aged 41, mining engineer, manager or general superintendent, 20 years' experience, 10 years in Mexico, 5 years Western States, 5 years abroad. Guarantee results, fluent Spanish, excellent references. New York interview. No. 282.

Member, aged 37, married, graduated mining engineer, 16 years' experience in gold, silver, lead, copper, and zinc mining and milling in the West, approximately as follows: Underground $4\frac{1}{2}$ years, stamp milling, concentrating and oil flotation $6\frac{1}{2}$ years, cyaniding 5 years, from shoveler and mill hand to general superintendent. Would consider starting in subordinate position if with prospects, on staff of a large mining concern. Would arrange for interview anywhere in North-eastern States, and will send best references on request. No. 283.

Operating engineer, aged 32, technical graduate, married, wants engagement. Have successfully operated coal and iron mines in United States and Canada. References furnished. No. 284.

Member, mining and metallurgical engineer, technical graduate, single, for the past three years employed as mill superintendent of a large gold-silver cyaniding plant in Spanish America will be open for position after May 1st, as mill superintendent or manager of a similar undertaking. Will go anywhere. Speaks Spanish and German. References furnished. No. 285.

Member, 37, geologist, familiar with mining work, graduate in mining from the Mass. Institute Technology, open for employment. 10 years' experience in applied geology. Now engaged in Montana and Idaho. No. 286.

Wanted: position as mining engineer or superintendent, by graduate mining engineer; 7 years' broad experience in mine operating and exploration work; until recently superintendent of large tonnage gold property; past record good; best references. No. 287.

Member, aged 23, technical graduate with experience in laboratory and non-ferrous foundry, as chemist, metallurgist, and executive; desires position offering advancement. No. 288.

Member, aged 24, technical graduate, with experience as chemist in acid and alkali manufacture; in iron and steel manufacture; in construction work, and as instructor in metallurgy is desirous of obtaining any of the following positions: research fellowship, subordinate position in iron and steel industry,—technical position. No. 289.

LOCAL SECTION NEWS

SOUTHERN CALIFORNIA SECTION

Executive Committee

SEELEY W. MUDD, *Chairman*

C. COLCOCK JONES, *Vice-Chairman*

FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

RALPH ARNOLD

A. B. CARPENTER

A. B. W. HODGES

WILLIAM F. STAUNTON

The third meeting of this winter season took place in a private dining room of the Sierra Madre Club in Los Angeles on Feb. 1, 1916, at 6:30 p.m., 22 members and guests being present. In the absence of the Chairman, S. W. Mudd, the Vice-Chairman, C. Colcock Jones, presided.

After an enjoyable dinner the meeting was called to order, and the first paper by Ralph Arnold on Conservation of Oil and Gas in the Americas was read by the Secretary. W. F. Staunton then read an important paper on The Influence of California Oil on the Mining Industry of the West. In this he touched upon the uses of oil as a fuel for generating power, especially in connection with the Diesel engine. This paper was discussed at some length by J. F. Mitchell-Roberts, R. B. Moran, F. H. Asbury and L. D. Godshall.

The Chair then introduced Robert B. Moran of the California State Mining Bureau, Deputy Supervisor of Gas and Oil, for the Los Angeles District, who addressed the meeting on the work of his department.

The meeting concluded with A. B. Carpenter's reading of a paper on The California Gasoline Industry by W. R. Hamilton, Manager of the Ventura Refining Co.

F. J. H. MERRILL, *Secretary*.

CHICAGO SECTION

CHARLES H. MACDOWELL, *Chairman*

LUTHER V. RICE, *Vice-Chairman*

H. W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History,
Chicago, Ill.

ALEXANDER K. HAMILTON

HENRY P. HOWLAND

GEORGE P. HULST

FREDERICK T. SNYDER

At a meeting of the Section on Mar. 9, 1916, at the Chicago Engineers' Club, the above given officers were elected. The attendance included members and guests of the Blast Furnace & Coke Association. After the dinner and business session, C. G. Atwater of the Barrett Co. gave an illustrated talk on "By products from Coal Distillation."

H. W. NICHOLS, *Secretary*.

MONTANA SECTION

J. L. BRUCE, *Chairman*W. C. SIDERFIN, *Vice-Chairman*M. H. GIDEL, *Secretary-Treasurer*, 1102 W. Galena St., Butte, Mont.

N. B. BRALY

W. T. BURNS

The annual meeting of the Montana Section was held at the Silver Bow Club, Butte, Mont., on Friday, Feb. 4, 1916, at 8:30 p.m.

An informal subscription dinner at \$1.50 a plate was enjoyed earlier in the evening by 49 members and guests.

After dinner, Chairman Smith called the meeting to order.

The minutes of the previous meeting, Oct. 8, 1915, were read and approved.

The reports of the Secretary-Treasurer for the last fiscal year were read and approved. These reports contained the information that the membership of the Section is now 177 and that there was such a favorable balance in the treasury that the Executive Committee had decided not to make requisition upon the parent Institute for an appropriation for 1916.

The following officers for the ensuing fiscal year were chosen by acclamation:

<i>Chairman,</i>	J. L. BRUCE,	Butte.
<i>Vice-Chairman,</i>	W. C. SIDERFIN,	Butte.
<i>Secretary-Treasurer,</i>	M. H. GIDEL,	Butte.
<i>Executive Committee,</i>	N. B. BRALY,	Butte.
<i>Executive Committee,</i>	W. T. BURNS,	Great Falls.

Under New Business, E. P. Mathewson gave a talk on Membership, supported by F. M. Smith.

J. L. Bruce talked upon the Naval Board Letter, which subject was discussed by E. P. Mathewson.

The members and guests then retired to the parlor of the Club where a technical session was held. J. L. Bruce acted as Chairman.

An interesting paper entitled Present Status of Oil and Gas Prospecting in Montana was presented by Prof. D. C. Bard, who is joint author of the paper with Chester Steele. The paper was illustrated by a large wall map. Discussion was participated in by Messrs. Mathewson, Laist, Billingsley, Bruce, Brophy, and Barker.

Frederick Laist gave an interesting talk on the metallurgy of zinc, reviewing briefly the retort method, and describing the new electrolytic process which he devised for the treatment of complex Butte ores. Those who discussed the subject were Messrs. Goodale, Smith, Bruce, Mathewson, Barker, and Brophy.

Upon proper motion, the society instructed the Secretary to extend to the officers of the Silver Bow Club its thanks for the use of the club on this occasion.

Upon proper motion, a vote of thanks was extended to the speakers of the evening, and also to the retiring officers of the Section, for their efforts during their term of office.

M. H. GIDEL, *Secretary*.

UTAH SECTION

C. W. WHITLEY, *Chairman*. WALTER FITCH, *Vice-Chairman*.
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave.,

E. R. ZALINSKI Salt Lake City, Utah. WILLIAM WRAITH

The semi-annual meeting of the Utah Section was held on Tuesday, March 7, 1916; 62 members went by special train from Salt Lake to the Arthur Plant at Garfield, and the Garfield Smelter, leaving Salt Lake at 1:00 p.m. and returning at 5:35 p.m. The officials of both the Utah Copper Co. and the Garfield Smelting Co. had representatives of their respective companies meet the members at the plants, and conduct them through, explaining the various processes.

At 7:30, 75 members attended a dinner at the Alta Club which was presided over by the Chairman, C. W. Whitley.

After the dinner, the Secretary read a letter from Institute headquarters, to which was attached copy of a letter from President Wilson, requesting each section to appoint a representative to act in conjunction with representatives from other engineering societies for the purpose of assisting the Naval Consulting Board in the work of collecting data for use in organizing the manufacturing resources of the country for the public service in the case of emergency. The Executive Committee presented the name of Lafayette Hanchett for this appointment and on motion duly seconded, this recommendation was ratified.

The Secretary then read a letter from President Richmond of the Salt Lake Commercial Club, requesting the appointment of a committee to act in conjunction with other committees to form a joint committee to assist in obtaining a Brigade Post at Fort Douglas, a Supply Depot in Salt Lake, and a Munitions Plant in Salt Lake.

After some discussion by Messrs. Wraith, C. W. Riter, and Leavell in the negative, R. C. Gemmill in the affirmative and W. M. Bradley, neutral, it was proposed (by Mr. Leavell), seconded and carried, that a committee consisting of the Executive Committee of the Section and Mr. Hanchett be appointed to investigate these matters and report to the Section.

For the purpose of aiding the increase of membership movement in the Institute C. H. Doolittle, *Chairman*, A. H. Richards, and C. W. Stimpson were appointed a committee to obtain the names of those who are eligible, but who are not yet members, and induce them to send in their applications for membership, Chairman Whitley asking all members present to render this Committee all assistance possible.

A paper entitled The Preparation of Powdered Coal for Reverberatory Furnaces was read by R. F. Barker, followed by a discussion, in which members participated. Mr. Whitley furnished the meeting further information on the subject. ERNEST GAYFORD, *Secretary*.

Resolved, that the members of the Utah Section of the American Institute of Mining Engineers extend their heartfelt sympathy and kindly wishes to Mrs. E. P. Jennings and family, in their recent great bereavement in the loss of husband and father. As a member of the Utah Section, Mr. Jennings was held in high esteem for his ability and character, and this feeling is shared by members throughout the country. His loss will be deeply felt.

And Be It Further Resolved, that a copy of this resolution be published in the Institute's *Bulletin* and be sent to the family of the deceased.

AFFILIATED STUDENT SOCIETY NOTES

The School of Mines Society of Columbia University was recognized as an affiliated Student Society by the Board of Directors of the Institute, at a meeting held on Feb. 15, 1916.

The School of Mines Society of the Oregon Agricultural College was recognized as an Affiliated Student Society of the Institute, at a meeting of the Board of Directors, held on Feb. 15, 1916.

The Minnesota School of Mines Society, Minneapolis, held a very successful and noteworthy meeting on Feb. 11, 1916, at which five of its members gave illustrated talks. The addresses were as follows: Working Underground in the Perseverance Mine, Treadwell, Alaska, by Sam Aronson; The Summer Trip of the Class of 1917 and Their Experiences on the Iron Ranges, by Edwin Sweetman; The Student in Butte and a Description of Conditions in This Great Mining Camp, by Hjalmar Abrahamson; The Washoe Reduction Works at Anaconda, with a Detailed Description of the Research Department, by Alvin T. Krogh; Some Mining Reminiscences of a Trip to Ecuador, by Guy A. Ingersoll. At the Feb. 23 meeting Dr. W. H. Emmons spoke on Geological Exploration. His talk was prefaced by a summary of the recent advances made on metallurgical, geological and mining lines and the need for intimate knowledge of these subjects on the part of the mining engineer. He then discussed the methods of the geologist in the field, illustrating his talk with views taken while on a trip through the Uncompahgre District of Wyoming for U. S. Geological Survey.

On Mar. 8, 1916, Horace V. Winchell spoke on Mining Law, explaining, with models and lantern slides, phases of various litigations; informal discussion ensued.

ADOLPH DOVRE, *President*.

The Mining and Metallurgical Club of the University of Toronto met on Feb. 14, 1916. Papers were read by Dr. Boyd, B. C. Tomlinson, and B. A. McCrodan.

B. A. McCRODAN, *President*.

The Mining Association, University of California has elected officers as follows:

<i>President,</i>	W. B. MILLER.
<i>Vice-President,</i>	M. K. KNOWLES.
<i>Treasurer,</i>	LEON GAZARIAN.
<i>Secretary,</i>	ROY STARBIRD.

Prof. A. C. Lawson has arranged for a series of lectures to be delivered by prominent mining men. T. A. Rickard and Frank H. Probert have addressed the Association, and Charles W. Merrill is next in order.

ROY STARBIRD, *Secretary*.

The Mining and Metallurgical Society, Lehigh University on Dec. 15, 1915, met with the other engineering societies of the University, Dr. H. S. Drinker presiding at the meeting.

R. S. Perry spoke at length on the popular topic of Preparedness, more particularly of The Function and Industrial Activities as Coefficients.

S. MARTIN, *President*.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from Sept. 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

- ANTHRACITE MINING LAWS OF PENNSYLVANIA, 1915. Harrisburg, 1915. (Gift of Penn State Library.)
- BITUMINOUS MINE LAW, PENNSYLVANIA, 1911. Harrisburg, 1911. (Gift of Penn State Library.)
- COLORADO INDUSTRIAL PLAN. By John D. Rockefeller, Jr. Including a copy of the plan of representation and agreement adopted at the coal and iron mines of the Colorado Fuel and Iron Co. 1916. (Gift of John D. Rockefeller, Jr.)
- GASOLINE MINE LOCOMOTIVES IN RELATION TO SAFETY AND HEALTH. U. S. Bureau of Mines, Bull. No. 74. Washington, 1915.
- HANDBOOK OF NATURAL GAS. Ed. 2. By H. P. Westcott. Erie, Pa., 1915.
- LEHRBUCH DER ELEKTROCHEMIE VON SVANTE ARRHENIUS. Leipzig, 1915.
- RECIENTES ADELANTOS EN LA PRÁCTICA METALÚRGICA EN EL PERÚ. By M. Fort. Lima, 1915. (Gift of Peru-Escuela de Ingenieros.)
- SHOT-FIRING IN COAL MINES BY ELECTRICITY CONTROLLED FROM OUTSIDE. Tech. Paper 108, U. S. Bureau of Mines. Washington, 1915.

Geology and Mineral Resources

IMPERIAL GEOLOGICAL SURVEY OF JAPAN. Topographical map, 1915.

ORIGIN OF THE ZINC AND LEAD DEPOSITS OF THE JOPLIN REGION, MISSOURI, KANSAS AND OKLAHOMA. Bull. No. 606, U. S. Geological Survey. Washington, 1915.

QUARRY MATERIALS OF NEW YORK—GRANITE, GNEISS, TRAP AND MARBLE. Bull. No. 181, N. Y. State Museum. Albany, 1916.

WEST VIRGINIA GEOLOGICAL SURVEY. County Reports. Detailed Report on Lewis and Gilmer Counties. By D. B. Reger. Morgantown, 1916.

(NOTE.—This volume contains 680 pages of text, illustrated; accompanied by a separate case of topographic and geologic maps covering the entire area in single sheets. Both of these counties lie within the coal, oil and gas belt of the State, and the economic geology and structural map of both should prove of great value and interest. Price of Report with case of maps, delivery charges paid by the Survey, \$2. Extra copies of geologic map \$1 each, and of the topographic map 50 c. each.

Military Engineering

ANNALS OF A FORTRESS. By. E. Violett-le Duc. Translated by Benjamin Bucknall. London, 1875.

MILITARY AEROPLANES. By G. C. Leening. San Diego, 1915.

Company Report

DEBEERS CONSOLIDATED MINES, LTD. Annual Report 27th Kimberley. 1915.

Trade Catalogues

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GENERAL ELECTRIC Co., Schenectady, N. Y. Bull. 43800, Incandescent Headlights for Street Railway Service. 1916.

NATIONAL TRANSIT PUMP AND MACHINE Co., Oil City, Pa. Bull. No. 9, The Foam System for Extinguishing Oil Fires.

SULLIVAN MACHINERY Co., Chicago, Ill. Bull. No. 70-A, Sullivan "Rotator" Hammer Drills. Jan., 1916.

GIFT OF AERONAUTICAL SOCIETY

Aeronautical Classics, edited for the council of the Aeronautical Society of Great Britain by J. O. B. Hubbard and J. H. Ledeboer. London, 1910-11.

Aston, W. G.—Model Flying Machines, Their Design and Construction. London. N. d.

Brockett, Paul.—Bibliography of Aeronautics. Washington, 1910.

Collins, F. A.—Second Boy's Book of Model Aeroplanes. New York, 1911.

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Flying, the Why and Wherefore by Aero Amateur. London. N. d.

Hering, D. W.—Essentials of Physics for College Students. New York, 1912.

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Langley, S. P.—Experiments in Aerodynamics. Ed. 2. Washington, 1902.

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Müller, Bruno.—Air Sacs of the Pigeon. Washington, 1908.

Peary, R. E.—Northward over the "Great Ice." 2 volumes. New York, 1898.

Petit, Robert.—How to Build an Aeroplane. London, 1910.

Richardson, H. C.—Hydromechanic Experiments with Flying Boat Hulls. Washington, 1914.

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- Conquest of the Air or the Advent of Aerial Navigation. New York, 1909.
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 Santos-Dumont, A.—My Air Ships. New York, 1904.
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 Verrill, A. H.—Gasoline Engines, Their Operation, Use and Care. New York, 1912.
 ———Harper's Air Craft Book. New York, 1913.

Periodicals

- AERIAL AGE. Vol. 1, Nos. 1, 4, 6, 7, 9, 11, 12; vol. 2, Nos. 1, 2.
 AERO. Vol. 2, Nos. 6, 20, 22; vol. 3, Nos. 1-7, 9-13, 17, 20, 24, 26; vol. 4, Nos. 1-2, 5, 7, 10-13.
 AERO (London). Vol. 5, Nos. 104, 105; vol. 6, Nos. 106-108.
 AERO AND HYDRO. Vol. 4, Nos. 14, 15, 17, 20; vol. 5, Nos. 13, 22, 24, 26; vol. 6, Nos. 1, 4, 6, 8-16, 18-23, 25-26; vol. 7, Nos. 1-26; vol. 8, Nos. 1-5, 7-17.
 AERO CLUB OF AMERICA. Bulletin, vol. 1, No. 4.
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 AERONAUTICS. Vol. 1, No. 5; vol. 13, complete.
 AERONAUTICS (London). Vol. 5, Nos. 47, 49; vol. 9, Nos. 114, 115.
 AIR SCOUT. Vol. 1, Nos. 1-5.
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 AMERICAN AERONAUT. Jan.-June, 1908; vol. 1, Nos. 1, 3.
 AMERICAN MATHEMATICAL SOCIETY. List of officers and members, 1915.
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 L'ASSOCIATION GÉNÉRALE AUTOMOBILE REVUE. Vol. 8, No. 2, 1909.
 AUTOMOBILIA. Vol. 5, no. 40.
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 FLY. Vol. 3, No. 12; vol. 4, Nos. 1-3.
 MOBILE ERA. Vol. 2, No. 5, 1912.
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- AMERICAN ROLLING MILL COMPANY. Story of Armco Iron Defeating Rust. Middletown, Ohio. N.d.
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 ———Burgess-Wright Aeroplane. Marblehead, Mass. N.d.
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 HESS-BRIGHT MANUFACTURING COMPANY. Ball Bearings in Flour and Feed Milling Machinery. Philadelphia. N.d.
 ———Hess-Bright. Philadelphia. N.d.
 NATIONAL CORRUGATED CULVERT MFG. CO. Making of Armco Iron. N.p. N.d.
 NEW DEPARTMENT MANUFACTURING CO. Bearing Friction and Its Elimination. Bristol, Conn. N.d.
 ROBERTS MOTOR COMPANY. Roberts Aviation Motor. Sandusky, O., n.d.
 STANDARD CO. Eagle Marine Engines. Torrington, Conn. N.d.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Feb. 10, 1916 to Mar. 10, 1916.

- ALLAN, FERGUS L., Min. Engr., Cons. Min. Engr., Mexico Mines of El Oro,
El Oro, Mex.
- ARMSTRONG, EMERSON WARREN, Met., Mina Bibilonia, La Libertad, Nicaragua, C. A.
- BABB, P. A., Blaisdell Coscotitlan Syndicate, Pachuca, Hgo., Mex.
- BANDY, PERCY J., Fundicion Delicias, Calle Delicias, Mexico, D. F.
- BELLWALD, J. N., Apartado 1488, Mexico, D. F.
- BILLICK, DON CARLOS, Min. Engr., Supt.,
Hess Gold Mines Co., Alturas, Modoc Co., Cal.
- BITLER, FRANK L., Cons. Min. Engr., Birkinbine Engineering Offices, Parkway Bldg.,
Broad and Cherry Sts., Philadelphia, Pa.
- BLAIR, ALBERT E., Gte. Gral. y Apoderado, Francisco Madero, San Pedro, Coah., Mex.
- BLANTON, EDWARD ANDERSON, JR., Prest., Blanton Copper Mining Syndicate,
Philadelphia, Pa.
- BORDWELL, EDWARD M., Mill Foreman, Nevada Hills Mining Co., Fairview, Nev.
- BULLOCK, STANLEY CROSSLAND, Min. Engr., 97 Bexley Road, Erith, Kent, England.
- CARTER, HERBERT F., 2a de San Augustin 53, Mexico, D. F.
- CLAUSEN, ERVIN H., Min. Engr.,
Asst. Mgr., Colorado Mining Co., Aroroy, Masbate, P. I.
- DENNY, GEORGE ALFRED, 411 Bancaria Bldg., 32 Cinco de Mayo, Mexico, D. F.
- DUNCAN, LINDSAY, Mech. Engr., Nevada Cons. Copper Co., McGill, Nev.
- EMPSON, J. B., Cons. Met. Engr., 3a Calle de San Augustin 78, Mexico, D. F.
- GALLAGHER, FRANCIS JAMES, Min. Engr., Oatman, Ariz.
- GIBBONS, FRANCIS J., Asst. Supt., Young Bros., Shultz, Ariz.
- GODIN, L. B., Care Fundicion de los Arcos, Almoloya de Alquisiras,
Est. de Mexico, Mex. Via Toluca.
- HINES, PIERRE ROSSITER, Min. Engr., Care W. M. Butler, First National Bank,
El Paso, Tex.
- HOYLE, CHARLES, Esperanza Min. Co., Apartado 8, El Oro, Mex.
- HUTCHINSON, JAMES WILLIAM, Genl. Mgr., Goldfield Cons. Mines Co., Goldfield, Nev.
- JOWETT, J. H., Genl. Sales Mgr., Ingersoll-Rand Co. and A. S. Cameron Steam
Pump Wks., 11 Broadway, New York, N. Y.
- LAIN, JOHN BYRNS, Mill Supt., Goldfield Cons. Mill. & Transportation Co.,
Goldfield, Nev.
- LANTZ, CARL A., Mining, Geol. Supt., Cia. de Santa Getrudis, S. A., and Cia.
Beneficiadora de Pachuca, S. A., Apartado No. 1, Pachuca, Hidalgo, Mex.
- LAWR, CLARENCE WILLIAM, Met. Chemist, Care Tigre Mining Co., Yzabal, Son., Mex.
- LEHMAN, GEORGE R., Min. Engr., Chief Engr., Inspiration Cons. Copper Co.,
Miami, Ariz.
- LOPEZ, ALEJANDRO, Medellin, Colombia, S. A.
- MCKINLAY, JAMES, 3a Calle de San Augustin 78, Mexico, D. F.
- MACFARLANE, THOMAS MURDOCH MARSHALL, Care Messrs. H. S. King & Co.,
65 Cornhill, London, E. C., England.
- MACNICHOL, ALEXANDER WILLIAM, Min. Engr., 1456 Fulton St., San Francisco, Cal.
- MARCH, ALBERT T., Supt., River Smelt. & Ref. Co., Keokuk, Ia.
- MORRISON, GEORGE A., Apartada 863, Mexico, D. F.
- NAKAGAWA, SHIN., Min. Engr., Imperial Bureau of Mines,
Dept. of Agriculture and Commerce, Tokyo, Japan.
- NORTON, HAROLD LEE, Engr., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
- NYBERG, EMIL, Vice-Pres., The Bolivia Gold Explor. Co., 207 Glencoe Bldg.,
Duluth, Minn.
- O'BRIEN, PATRICK C. K., Hotel Essex, 684 Larkin St., San Francisco, Cal.
- PANYITY, L. S., Geol., Ohio Fuel Supply Co., Columbus, O.
- PARKER, FRANK W., Supt., Commercial Min. Co., Rainbow Mine, Rye Valley, Ore.
- PENN, FREDERICK H., General Foreman, Belmont Milling Co., Tonopah, Nev.
- PLATT, JAMES M., Supt., Guadalupe Arcos, Zacualpan, Mex.
- RAHILLY, HAROLD J., U. S. Bureau of Mines, Pittsburgh, Pa.
- ROBERTS, EDWARD J., Prest. and Mgr., Daley Cons. Mines Co., 27 West 3rd St.,
Salt Lake City, Utah.

SANDERS, W. MURRAY, Asst. General Mgr., Duquesne Mining & Reduction Co.,
Duquesne, Ariz.
SAUERSCHNIG, JOSE Calle J. M. Pino de Saurez 37, Mexico, D. F.
SAXMAN, PETER MARSEILLES Supt., The Ebensburg Coal Co., Colver, Pa.
SIMPSON, W. EVAN, Care Denny Bros., Bancaria Bldg., 32 Cinco de Mayo,
Mexico, D. F.
STEPHENSON, EUGENE AUSTIN, Instructor in Geol. Univ. of Chicago, Chicago, Ill.
STREVEVS, JOHN EDWARD LLOYD, Tech. Chem., Broken Hill Chambers,
31 Queen Street, Melbourne, Vic., Aust.
SULTZER, H. D., Min. Engr., Asst. Engr., Anaconda Copper Min. Co., Butte, Mont.
SWEENT, MICHAEL J. Pres. and Genl. Mgr., Rex Cons. Mining Co., Wallace, Ida.
THOMPSON, PERCY WILLIAM, Oil Engr., Asst. Supt., Standard Oil Co., Fullerton, Cal.
TRAGER, GUSTAV E., Los Arcos Smelt. & Min. Co., Almoloya de Algecires,
Los Arcos, Mex.
UNDERWOOD, JERROLD ROSCOE Mine Owner and Operator, Granby, Mo.
WEED, ROBERT C., Met., Minnesota Steel Co., Morgan Park Sta., Duluth, Minn.
WESTALL, ALFRED HELEM, Genl. Supt., Nevada Hills Min. Co., Fairview,
Churchill Co., Nev.

Associate Members

HAYLER, GEORGE E., JR. Genl. Mgr., The Empire District Elec. Co., Joplin, Mo.
LACAUD, R. AMILIEN 2a de San Augustin 53, Mexico, D. F.
ROBERTS, B. J. Secy., Deister Miners Supply Co., Fort Wayne, Ind.
ZAVALA, ENRIQUE 3a de San Augustin 73, Mexico, D. F.

Junior Members

BARKER, LYLE MAXON Missouri School of Mines, Rolla, Mo.
CASTILLO, RICARDO E. 482 Birkel Ave., So. Bethlehem, Pa.
CHEN, FAN Box 734, Golden, Colo.
DENNIS, PAUL J. 316 Quincy St., Rapid City, S. D.
DOWD, JAMES J. Box 10, Rolla, Mo.
DOYLE, JOHN JOSEPH, Chief Clerk, Missouri Bureau of Geology and Mines,
Rolla, Mo.
HALL, CLARENCE K. 140 N. Barnard St., State College, Pa.
HILL, WILLIAM J. 1003 W. Broadway, Butte, Mont.
HOFFMANN, JOHN, S. Missouri School of Mines, Rolla, Mo.
HOGAN, EUGENE 801 N. Montana St., Butte, Mont.
HOUSHOLDER, E. ROSS Box 304, Rolla, Mo.
LIND, A. LESLIE 101 S. Crystal St., Butte, Mont.
OLIVEROS, R. P., Assayer, Chemist and Engr., Atlas Min. & Mill. Co., Sneffels,
Ouray Co., Col.
STEPHENSON, FRANCIS LEWIS 77 Church St., Bethlehem, Pa.
STRANG, WILLIS B. 815 West Galena St., Butte, Mont.
UDE, GEORGE EDGAR Missouri School of Mines, Rolla, Mo.
VOGEL, HERMAN H. Missouri School of Mines, Rolla, Mo.
WHITE, HAROLD EDWARD Psi Upsilon House, Lehigh Univ., So. Bethlehem, Pa.
WILLIAMS, FRED P. 28 West 46th St., New York, N. Y.
Total Membership, Mar. 10, 1916. 5,400

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period Feb. 10, 1916 to Mar. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A

sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

William S. Adams, Anaconda, Mont.

Proposed by Selden S. Rodgers, Frederick Laist, Albert E. Wiggin.

Born 1883, St. Louis, Mo. 1890-97, Grade Schools, St. Louis, Mo. 1899-1902, Lowell High, San Francisco, Cal. 1902-06, Grad., Univ. of California. 1906-07, Washing foreman, Tremont Coal and Coke Co., Wilkeson, Wash. 1907-09, Cyanide operator, North Star Mining Co., Grass Valley, Cal. 1909-11, Mill Supt., Darby and Reduction Co., Mazuma, Nev. 1914-16, Slimes plant foreman, Anaconda Copper Mining Co., Anaconda, Mont.

Present position: Filtration Engr., Anaconda Copper Mining Co.

James Elanson Alley, Tonopah, Nev.

Proposed by Frederick Bradshaw, E. A. Julian, A. H. Jones.

Born 1888, Mountain Home, Ida. 1893-96, Public School, Caldwell, Ida. 1896-1901, Public School, Salt Lake City, Utah. 1901-05, Salt Lake City, Utah. 1905-10, Utah State School of Mines, B. S. in Min. Engrg. 1910-11, Mill Supt., George F. Roth Mining Co., Neal, Ida. 1911-12, Mine Foreman, Columbus Extension Min. Co., Alta, Utah. About twelve months in various places of minor jobs, including, battery man, tableman, machine man, etc., Belmont Mill. Co.

Present position: 1912 to date; Shift Foreman, Belmont Mill. Co.

Charles Welling Badgley, El Paso, Tex.

Proposed by Kuno Doerr, James Heggie, Alan F. McCormick.

Born 1885, Milwaukee, Wis. 1902, Denver Public Schools. 1902-06, Colorado School of Mines, E. M. 1906-08, Asst. Chem., Globe Plant. 1909, Chief Chem., American Smelt. & Ref. Co. 1910, Chem. and Mill Supt., Western Chemical Works, Denver. 1911-12, Engr. and Chem., Standard Chemical Co. (uranium and vanadium fields in and around Paradox Valley, Col.). 1912-13, Asst. in Research Laboratory, American Smelting & Ref. Co., Perth Amboy, N. J.

Present position: Research Chem., El Paso Smelt. Works.

Adolf G. Ballenberg, Chicago, Ill.

Proposed by W. E. Hopper, F. W. McNair, F. W. Sperr.

Born 1893, Chicago. 1912, Grad., Univ. High School, Chicago. 1915, Grad., Michigan College of Mines, Houghton, Mich., B. S. and E. M. 1915 to date; Engr., Cobwell Corporation, New Bedford Extractor Co., New Bedford, Mass. 1916 to date; Engr., Porcupine Exploration Co., Ontonagon, Mich.

Present position: As above.

Edward Ludlam Blossom, New York, N. Y.

Proposed by Lawrence Addicks, J. E. Johnson, Jr., Bradley Stoughton.

Born 1867, Brooklyn, N. Y. 1888, Grad., Harvard Univ., A. B. 1898-99, Special studies in Chemistry and Metallurgy, Michigan College of Mines. 1888-95, Commercial business. 1895-96, Mgr., Sonora Copper Co., La Puertitas, Son. 1896-97, Smelter clerk, Montana Ore Purchasing Co., Butte; and B. C. Smelting and Refining Co., Trail, B. C. 1898, Supt. Smelter, B. C. Smelt. & Ref. Co. 1901-04, Supt., Smelter, Montana Ore Purchasing Co. 1904-13, Scouting and Consultation Work. 1913-15, Phelps Dodge & Co., Leaching experiment.

Present position: Staff of Research Corpn.

Eugene Harding Broughton, Warren, Ariz.

Proposed by Charles A. Mitke, G. F. G. Sherman, Roger T. Pelton.

Born 1887, Jefferson City, Mo. 1901, Grad., Grade Schools, St. Joseph, Mo. 1905, High School, St. Joseph, Mo. 1912, Missouri School of Mines, B. S. 1906, Rodman, Central R. R. of Louisiana, Southern Kansas City. 1907-08, Engr., Red Carbonate Explor. (Cal. & Ariz. Min. Co.) Twin Buttes, Ariz. 1908, Asst. Locating Engr., Cal. & Ariz. Min. Co., Mammoth, Ariz. 1910, summer, Asst. City Engr., Jefferson City, Mo. 1912-13, Engr., Courtland Explor. (Cal. & Ariz. Min. Co.) Courtland, Ariz. 1914, Transitman, Copper Queen Cons. Min. Co., Bisbee, Ariz.

Present position: Experimental Engr., Copper Queen, Bisbee.

Thomas Ellis Brown, Jr., Morristown, N. J.

Proposed by Joseph Struthers, R. V. Norris, Thomas E. Brown.

Born 1889, Yonkers, N. Y. 1907-12, School of Mines, Columbia Univ., E. M. 1912, Griscom Russell Co. 1912-13, American Bridge Co.

Present position: 1913 to date; Asst. to Thomas E. Brown.

Samuel W. Burford, Dubuque, Ia.

Proposed by J. H. Janeway, Arthur Thacher, H. A. Wheeler.

Born 1880, Pittsburgh, Pa. 1896-1900, Pittsburgh High School. Remainder of education in the field. 1900-01, Chainman and Instrument man, Pittsburgh Coal Co. 1901-03, Div. Engr. and Chief Draftsman, Monongahela River Cons. Coal and Coke Co. 1903-06, Chief Engr., Pittsburgh & Buffalo Co. 1906-07, Min. Salesman, Aultman & Taylor. 1907-09, Eastern Div., Water Tube and Power Plant Salesman, Atlas Engine Works.

Present position: 1909 to date; Genl. Mgr., Cleveland Min. Co., Lawrence Mines Co.

George Bruce Butterfield, Greensburg, Pa.

Proposed by Milton R. Evans, E. C. Lee, H. M. Wilson.

Born 1890, Greensburg, Pa. 1909-13, Pennsylvania State College. 1907-12, summers, Jamison Coal and Coke Co. 1913-15, Jamison Coal and Coke Co.

Present position: 1915 to date; Senior Inspector, The Associated Co.

Wallace H. Christie, San Francisco, Cal.

Proposed by Charles Janin, Edward H. Benjamin, Newton Cleaveland.

Born 1860, Atlanta, Ill. 1885-1896, Black Hills, Utah, Wyoming for own account. 1896-1902, Oroville field with Thos. Couch and for self in exploration work. 1903-11, Associated with W. P. Hammon in exploration work, and for own account.

Present position: 1912 to date; Vice-Pres. and Mgr., Alta Bert Gold Dredging Co.

Clifford Corp, Globe, Ariz.

Proposed by P. G. Beckett, L. O. Howard, I. H. Barkdoll.

Born 1882, Nickerson, Kans. 1904, Nickerson College, Nickerson, Kans. 1908, Univ. of Kansas, B. S. in Mech. Engrg. 1899-1903, Mechanic, Shops. 1909, Constr. Engr., power plant and mechanical laboratories, Univ. of Kansas. 1910, Engr., Riverside Engine Co., National Transit Co., Oil City, Pa. 1911, Draftsman, Calumet & Arizona Copper Co., Douglas, Ariz. 1912, Draftsman, Miami Copper Co.

Present position: Draftsman, drawing and design, Old Dominion Copper Mining & Smelting Co.

Arthur I. D'Arcy, Goldfield, Nev.

Proposed by K. M. Simpson, George H. Garrey, R. H. Downer.

Born 1879, Denver, Col. 1903, Missouri School of Mines, M. E., Rolla, Mo. 1903-05, Field Engr., Guggenheim Explor. Co. 1906, Mgr., Denver Mine, Rhyolite, Nev. 1907-08, Mgr., Jack Pot Mine, Wonder, Nev. 1908-11, Operation of leases and mining properties owned personally. 1911-14, Field Engr. for George Wingfield.

Present position: 1914 to date; Genl. Mgr., Reorganized Booth Min. Co., Reorganized Kewanas Min. Co., Atlanta Mines Co.

John Davenport, Brighton, Mass.

Proposed by W. G. Matteson, Charles H. White, L. C. Craton.

Born 1888, Allston, Mass. 1903-06, Mechanic Arts High School, Boston, Mass. 1907-09, Harvard College, Cambridge, Mass. 1909-12, Colorado School of Mines, E. M., Golden, Col. 1912, solution man for John Cross, Tuscarora, Nev. 1912-14, Mexican Candelaria Co., S. A., San Dimas, Durango, Mex. 1914 to date, United Mineral Co., Boston, Mass.

Present position: Supt., United Mineral Co.

L. G. Duncan, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1885, Kansas City, Mo. 1906, Michigan College of Mines, B. S., E. M. 1906, Asst. Engr., Shattuck-Arizona Copper Co. 1907, Mine Examination, Nevada Smelter and Mines Corp'n. 1909-10, U. S. Forest Service, Cal. 1910-12, Reclamation Work, San Simon, Ariz.

Present position: 1912 to date; Oxide and Pottery Dept., Mineral Point Zinc Co.

Louis D. Fry, Bisbee, Ariz.

Proposed by Roger T. Pelton, Arthur Notman, LeRoy B. Mitchell.

Born 1880, St. Louis, Mo. 1903, Grad., Colorado State School of Mines, E. M. 1903-06, Assayer, Surveyor, with various small companies in Chih. old Mexico. 1906-08, Supt. small smelter in Guerrero, Mex. 1908-14, Supt. lead smelter, Mazapil Copper Co., Saltillo, Coah., Mex.

Present position: 1914 to date; Engrg. Dept., Copper Queen Min. Co.

Author Hendrix Gill, Perth Amboy, N. J.

Proposed by H. H. Alexander, Clarence P. Linville, Willard S. Morse.

Born 1878, Boring, Md. 1896-1900, Grad., Lehigh Univ., M. E. 1900-01, Schenectady Loco. Works. 1901-03, Prof., Mech. Engrg. and Physic. Grove City College. 1903-06, Instr., Univ. of Pennsylvania, Philadelphia, Pa. 1906-08, Engrg. office, Amer. Smelt. & Ref. Co., Denver, Col. 1908-09, Supt., Power Plant, Steptoe Smelt. & Min. Co. 1909-11, Asso. Prof., Mech. Engrg., Pennsylvania State College, State College, Pa.

Present position: 1911 to date; Supt., Perth Amboy Plant, Amer. Smelt. & Ref. Co.

F. A. Goodale, You Bet, Cal.

Proposed by Y. C. Kuang, Harry J. Wolf, A. J. Hoskin.

Born 1883, Centralia, Ill. 1901, Grad., Trinidad High School, Colo. 1910, Grad., Colorado School of Mines, E. M. 1902, Asst. Engr., Colorado Fuel and Iron Co. 1904, Asst. Engr., Santa Fe R. R. 1906, Special examination work, Nevada Douglas Copper Co. 1906-09, Member, Goodale & Koerner Engineering Co., Buckskin, Yerington and Rawhide, Nev. U. S. Mineral Surveying, Assaying and General Engineering. 1910-15, Chief Engr., Delta Land Co. H. Lain Farmer Escalante Irrigation Co., Delta, Col. Also leasing and general engineering work in Idaho Springs, Col.

Present position: 1915 to date; Agent, You Bet Mining Co. and Engineer, Chinese Min. Co.

D. S. Grenfell, Depue, Ill.

Proposed by G. S. Brooks, George C. Stone, J. H. Janeway.

Born 1892, Black Earth, Wis. 1914, Univ. of Wis. Ch. E. 1914-15, Research Asst., Univ. of Wis. 1915-16, Chem., American Zinc, Lead and Smelt. Co., Caney, Kans.

Present position: Chem., Mineral Point Zinc Co.

George Gary Griswold, Maurer, N. J.

Proposed by H. H. Alexander, Clarence P. Linville, F. W. Traphagen.

Born 1866, Erie, Pa. 1882-86, E. Denver High School. 1892-96, Col. State School of Mines. 1896-1900, Phila. S. & R. Co., Pueblo, Col. 1900-05, A. S. & R. Co., Globe Plant, Denver, Col. 1905-07, Ohio and Col. S. & R. Co., Salida, Col. 1907-10, Leasing and Mining. 1910, 1912-16, A. S. & R. Co., Maurer, N. J.

Present position: Supt., Smelting Dept. and Lead Refinery, A. S. & R. Co.

John B. Guernsey, New York, N. Y.

Proposed by F. U. Humbert, Frank Lyman, Benjamin B. Lawrence.

Born 1886, Elmira, N. Y. 1907-11, Grad., New York Univ. 1906-07, Branch organization work under Controller's Dept., Underwood Typewriter Co., New York. 1911 to date, Industrial reorganization work on own responsibility, independent of any engineering firms or companies. Experience in reorganization work varied but principal work has been in manufacture of news print paper, wood pulp, cotton waste and cotton print, and pig iron, and mining of coal and iron ore.

Present position: Genl. Mgr., The Low Moor Iron Co. of Va.

Royal Sheppard Handy, Kellogg, Ida.

Proposed by Stanley A. Easton, W. McM. Huff, L. K. Armstrong, J. McD. Porter.

Born 1880, Loveland, Col. 1888-91, Public Schools, Fort Collins, Col. 1895, Public School, Nampa, Ida. 1901, Univ. of Cal. 1895, Rodman, Boise, Nampa & Owyhee Ry. Co., Nampa, Ida. 1896-97, Carman, Surveyor's Assistant, Black Jack Min. Co., Silver City, Ida. 1898, Bookkeeper & Assayer, Poorman Gold Mines, Ltd., Silver City, Ida. 1899, Transitman & Field draftsman, Boise Nampa & Owyhee Ry. Co. 1900, Bookkeeper & Surveyor, Virtue Cons. Mines, Silver City, Ida. 1902, Assayer, Bookkeeper, Surveyor, Red Boy Cons. Min. Co., Granite, Ore. 1904, Assayer, Bookkeeper, Surveyor, Angel Quartz Min. Co., Angels Camp, Cal. 1905, Smelter Representative, Yard Foreman, Bunker Hill & Sullivan M. & C. Co.

Present position: 1909 to date; Mill Supt., Bunker Hill & Sullivan M. & C. Co.

Purl L. Harms, Lead, S. D.

Proposed by W. J. Sharwood, A. A. Anderson, A. J. M. Ross.

Born 1885, Platteville, Wis. Until 1901, State Normal School, Platteville, Wis. 1902, Grad., Platteville Business College, Platteville, Wis. 1914, Grad., Wisconsin School of Mines, E. M. 1905-08, prospecting for Galena and Smithsonite with L. L. Harms, Platteville, Wis. 1909-10, Platteville Lead and Zinc Co., L. L. Harms, President, in zinc roaster and in lead and zinc mill.

Present position: Mine Sampler, Homestake Mining Co.

James Harbison Hensley, Jr., Miami, Ariz.

Proposed by B. Britton Gottsberger, F. W. MacLennan, H. P. Bowen.

Born 1884, Socorro, N. Mex. To 1902, Common school education in public schools of Col. 1902-06, Colorado School of Mines, E. M. 1906, Assayer, Majestic Copper Min. Co., Milford, Utah. 1907, various positions with The Alaska Smelt. & Ref. Co., Hadley, Alaska. 1908, Engr. on construction, Nevada-California Power Co., Goldfield, Nev. 1909, Min. Engr., with M. O. Danford, Trinidad, Col.

Present position: 1909 to date; Min. Engr., Miami Copper Co.

Oscar H. Hershey, Kellogg, Ida.

Proposed by Stanley A. Easton, Fred W. Callaway, L. K. Armstrong, J. McD. Porter.

Born 1874, Blue Rock, Pa. 1890, High School, Lebanon, Pa. 1908, Bunker Hill & Sullivan Mining & Concentrating Co. 1912, Member of engineering firm, Burch, Caetani & Hershey. 1913, Mountain Copper Co.

Present position: General practice as Min. Geol.

Robert Hoffman, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1871, Chemnitz, Germany. 1887-89, German High School. 1889-1892, Mechanical work in Germany. 1892-94, Mechanical work through U. S. 1894-1905, Asst. Foreman, Machine Shop, Matthiessen & Hezler Zinc Co., La Salle, Ill.

Present position: 1905 to date; Chief of Service Dept., Mineral Point Zinc Co.

C. Vey Holman, Rockland, Me.

Proposed by Charles F. Rand, William F. Meredith, Joseph W. Richards.

Born 1861, Poughkeepsie, N. Y. 1878-1882, Harvard College. 1875-78, Boston Public Latin School. 1882, Boston Univ. Law School. 1900-02, Univ. of Maine Law School, LL. B.; 1903, LL. M. 1902-05, Lecturer on Mining Law in Boston Univ. 1902-04, Lecturer on Mining Law, Univ. of Maine. 1910-11, State Geol. of Maine. 1908-10, Pres. of Caribou Gold Mines, operating the gold mines of the Caribou Gold District, Nova Scotia. 1907-16, Owner and developer of Holman Molybdenite Mine, Catharine Hill, Me. Life Member, Association of Harvard Engineers.

Present position: Mgr. and Owner, Caribou Gold Mines and Holman Molybdenite.

William R. Jarvis, Pittsburgh, Pa.

Proposed by Howard N. Eavenson, Samuel A. Taylor, Frederick K. Copeland.

Born 1871, Claremont, N. H. 1893, Grad., Dartmouth College. Since 1899, Sullivan Machinery Co. 1899-1901, In Chicago office. 1901-05, in Michigan and Minnesota as Mgr. for that territory. Iron and Copper Mines. 1905 to date, Mgr. of Pittsburgh and West Va. territory, Sales and Engineering work in Coal Mines and Quarries.

Present position: District Mgr., Sullivan Machinery Co. of Chicago.

Thomas B. Jones, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1880, Platteville, Wis. 1909, Platteville Normal School. 1906, Prospecting Zinc Ore, Platteville, Wis. 1909-10, Supt., Platteville Gas Co. 1910-12, Mineral Point Zinc Co.

Present position: Asst. Chief Spelter, Mineral Point Zinc Co.

W. M. Kelsey, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1881, Salamanca, N. Y. 1905, Cornell, A. B. 1906 to date, Mineral Point Zinc.

Present position: Genl. Supt., Mineral Point Zinc Co.

Arthur M. Kivari, Victor, Col.

Proposed by Irving T. Synder, H. P. Nagel, Jr., Justin H. Haynes.

Born 1892, Calumet, Mich. 1909, Grad., Hancock High School, Mich. 1910, Freshman, Univ. of Mich. 1911-14, Michigan College of Mines. 1911, Underground laborer, Copper Range Cons. Min. Co., Painesdale, Mich. 1911-12, Surveyor in Michigan. 1912, Lake Superior-Ophir G. M. Co., Ophir, Colo. 1912-13, Millman, Ophir Gold Mines, Ophir, Colo. 1913-14, Engr., Junta Cons. G. M. Co., Telluride, Col. 1914, Millman, Primos Chem. Co., Vanadium, Col. 1915, Assayer, Cordova Mines, Ltd., Cordova Mines, Can. 1915, Assayer and Millman, Colorado and Nevada M. & M. Co., Nelson, Nev.

Present position: 1915 to date; Mill draftsman, Vindicator Cons. G. M. Co.

G. N. Le Roux, Salt Lake City, Utah.

Proposed by Ernest Gayford, C. W. Stimpson, Morris P. Kirk.

Born 1873, Pass Christian, Miss. General Mining, Mill, Smelter, etc., experience during past six years. 1895-99, George B. Carpenter & Co., Chicago. 1899-1902, Republic Rubber Co., Youngstown, O. 1902-10, The B. F. Goodrich Co., Akron, O. 1910-12, N. Y. Leather Belting Co., New York. 1913, Manhattan Rubber Mfg. Co.

Present position: Genl. Sales Agent, The Manhattan Rubber Mfg. Co., Passaic, N. J.

Donald M. Liddell, Elizabeth, N. J.

Proposed by Walter Renton Ingalls, Burr A. Robinson, Allen H. Rogers.

Born 1879, Lawrenceburg, Ind. 1897-1900, John Hopkins Univ., Baltimore, Md., B. A. 1900-01, Chemist and Assayer, Detroit Copper Min. Co., Morenci, Ariz. 1901-05, Asst. Assayer, Chief Assayer, Blast Furnace Supt., General Engr. Staff, Baltimore Copper Smelting and Rolling Co., Baltimore, Md. 1905-08, Blast Furnace Supt., later in charge of byproducts recovery, sampling and statistical work, U. S. Metals Ref. Co., Chrome, N. J. 1908-07, Retained by Tom Cobb King as Consultant on his electrolytic nickel experiments. 1908-10, Cashier & Ch. Clerk, U. S. Metals Refg. Co., Grasselli, Ind.

Present position: 1910 to date; Editorial staff, associate editor, managing editor, Engineering and Mining Journal, New York.

Horace Tharp Mann, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1881, Gallatin, Mo. 1904-08, Missouri School of Mines, B. S. in Mining; 1909, M. S.; 1910, E. M. 1900-04, Assayer and Chemist, U. S. Smelt. Co., Canon City, Col. 1908, Engr., Alvarado Gold Min. Co., Congress Junction, Ariz. 1908, Instr., Metallurgy and Ore Dressing, Missouri School of Mines. 1913, Asst. Prof., Metallurgy and Ore Dressing.

Present position: 1914 to date; Associate Prof., Metallurgy and Ore Dressing in charge of Department.

Charles Mendelsohn, Globe, Ariz.

Proposed by C. Legrand, L. O. Howard, P. G. Beckett.

Born 1884, New York City. 1890-99, Public Schools, Boston, Mass. 1901-05, Grad., Harvard Univ., S. B. 1905, Miner; 1905-08, Mining and Structural engineering work, Homestake Mining Co., Lead, S. D. 1909-10, Engr., El Tiro Copper Co., Silverbell, Ariz. 1910-12, Mechanical and structural draftsman, Miami Copper Co.

Present position: 1912 to date; Mech. Engr., Old Dominion Copper Mining and Smelt. Co.

C. C. Nitchie, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1881, Evanston, Ill. 1905, Cornell, A. B.

Present position: 1905 to date; Chem. Engr., Mineral Point Zinc Co.

Israel O. Proctor, Butte, Mont.

Proposed by William N. Rossberg, D. C. Bard, C. H. Bowman.

Born 1884, Montana. 1903, Grad., Great Falls High School, Great Falls, Mont. 1904-08, Montana State School of Mines, Butte, Mont., E. M. 1908-12, Field Engr., Globe Mines Explor. Co. During this time was active in operation of hydraulic mining gravel in Idaho; Associated with Mr. D. C. Bard. 1912-14, Engr., Colusa Parrot Ore Testing Plant. 1914-15, Mill Supt., Bear Gulch, Mont. for Messrs. Bulinberg & Heggins of Deer Lodge, Mont.

Present position: Mill Testing and Metallurgical Testing, Timber Butte Milling Co.

Norman Hatfield Read, Manchester, Mass.

Proposed by Howard W. DuBois, Robert Peele, Robert M. Raymond.

Born 1891, Boston, Mass. 1909-13, Yale, B. A. 1915-16, Columbia University. 1912, Geol. Dept., Eclipse Oil Co., Casper, Wyo. 1913, with M. B. Burke in mine examinations, Cripple Creek, Col. 1914, Drilling and Testing Placer Ground, Parker-Alaskan Gold Co., Kenai Peninsula, Alaska. Examination and report on the Gilpatrick Mine, Moose Pass, Kenai Peninsula, Alaska, with other investigations of both lode and placer prospects on the Kenai Peninsula.

Present position: Engaged in graduate work in Ore Deposits and Geology, Columbia Univ.

D. D. Robbins, Depue, Ill.

Proposed by George Sage Brooks, George C. Stone, J. H. Janeway.

Born 1882, Louisville, Ky. 1909, New Mexico School of Mines, E. M. 1909-10, Chem., Chino Copper Co. 1910, Engr. and Mill Supt., Germania Min. Co. 1910, Chief Sampler, Inspiration Copper Co. 1911, Mech. Foreman, Universal Portland Cement Co. 1912, Draftsman, Mineral Point Zinc Co.

Present position: 1912 to date; Chief Ore and Testing Dept., Mineral Point Zinc Co.

Arthur E. Sargent, Schumacher, Ont., Canada.

Proposed by Luther B. Eames, Maurice W. Summerhayes, C. H. Poirier.

Born 1885, Bridgewater, S. D. To 1903, Rapid City, S. D., public schools. 1903-05, Grad., S. D. State School of Mines, Commercial Dept. 1906-07 and 1909-10, S. D. School of Mines, B. S. 1906, summer, Assayer, Mogul Min. Co., Pluma, S. D. 1908, Engr., East Lake Min. and Mill Co., Edwards, Col. and Columbus Cons. Min. Co., Alta, Utah. 1909, summer, Engr., Ivanhoe Gold Min. Co., Custer, S. D. 1909-13, Assayer, Engr., and Met., Lundberg, Dorr and Wilson, Terry, S. D. 1913-14, Engr., Construction, Mogul Min. Co., Terry, S. D.

Present position: 1914 to date; Mill Supt., Porcupine Vipond Mines, Ltd.

Samuel Shapira, New York, N. Y.

Proposed by W. B. Devereux, Jr., J. H. Devereux, H. P. Henderson.

Born 1884, Wilkes-Barre, Pa. 1898-1901, Boston English High School, Boston, Mass. 1901-04; 1909-10, Mass. Inst. of Tech., Boston, Mass., B. S. 1904, summer in Colorado—Student of mining operations. 1905, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1907-11, Min. Engr., Witherbee Sherman & Co., Mineville, N. Y. 1912, private practice, 50 Church St., New York, N. Y. 1912-14, Designing and Metallurgical Engr., Empire Steel & Iron Co., Mt. Hope, N. J. 1914-16, Min. Engr., Mines Management Co., New York, N. Y.

Present position: Min. Engr., Wilkens and Devereux.

William Sharp, Goldfield, Nev.

Proposed by K. M. Simpson, George H. Garrey, R. H. Downer.

Born 1885, Vernon, Utah. 1906, Univ. of Utah, B. S. 1906-10, Assaying and Surveying, Richmond-Eureka Mining Co., Eureka, Nev. 1910, Assaying and Surveying, Buckhorn Mines Co., Buckhorn, Nev. 1911-12, Assaying and Engineering, Nevada Hills Mining Co., Fairview, Nev.

Present position: 1912 to date; Asst. to A. I. D'Arcy, Genl. Mgr., Wingfield Outside Properties.

Eugene Sherwood Sheffield, Peking, China.

Proposed by F. L. Cole, T. A. Rickard, Frank H. Probert.

Born 1881, Santa Barbara, Cal. 1900-04, Univ. of California College of Mining, B. S. 1904, common miner, mill man and assayer, Treadwell Min. Co. 1906, Field Engr., Bradley, MacKenzie & Requa. 1909, Assayer and Cyanide Foreman, Buster Mining Co., Ida. 1909, Driller and later Field Mgr., Henderson Oil Co., Cal. 1910, Supt. of Production, North American Oil Consol. 1913, Field Supt., General Petroleum Co. 1914, Field Supt., Standard Oil Co. of N. Y.

Present position: Asst. Mgr., Producing Dept., Standard Oil Co. of N. Y.

Karl Frederick Smith, New York, N. Y.

Proposed by William Campbell, Siegfried Fischer, Jr., James T. Kemp.

Born 1885, New York, N. Y. 1902-04, Drury Academy. 1904-08, U. S. Naval Academy. 1914-15, Post Graduate School, U. S. N. A. 1908-16, Engineering duty, Navy. 1912, Survey duty. 1915, Grad. School U. S. N. A. and Naval Experiment Station.

Present position: Lieutenant, U. S. Navy.

W. R. Smith, Mineral Point, Wis.

Proposed by G. S. Brooks, A. D. Beers, J. H. Janeway.

Born 1864, Mineral Point, Wis. 1884, to date; Mineral Point Zinc Co.

Present position: Supt., Mineral Point Zinc Co.

Henry John Sommercamp, Payette, Ida.

Proposed by Robert Rhea Goodrich, E. K. Soper, Francis Jenkins.

Born 1869, Silver City, Ida. Public School. Business College, Chicago, Ill.
Assaying and Chemical Analyses under Prof. Hirshing, Salt Lake City.

Frederick Spengler, Seattle, Wash.

Proposed by Amos Slater, James A. Kelly, I. F. Laucks.

Born 1879, St. Louis, Mo. Education, general, in public and other schools. 1895-99, Technical, St. Matthews, San Mateo, Cal. 1899-12, Univ. of Cal., College of Mines, Berkeley, Cal. 1896-02, Assistant in Assay Office, Victoria, for about two years during time spent out of school between 1896 and 1902 making fire assays and wet assays. 1902-04, Mech. and Elec. Dept., Louisiana Purchase Exposition Co., St. Louis. 1904-05, U. S. Eng. Dept., Louisville, Ky. District office at determining feasibility of 9 ft. channel maintenance. 1906, Private practice of general engineering and construction with Charles E. Young, St. Louis. 1909-15, variously employed in British Columbia, Idaho, and Washington; Engr., Twin Lakes Land Irrigation Co., Twin Lakes, Ida.; Asst. to City Engr., Victoria, B. C.; Engr., Carbon Coal & Clay Co., Bayne, Wash.; Engr., Big 6 Mine, near Palmer, Wash.; Asst. to Amos Slater, Seattle.

Present position: Engr., Western Exploration Co.

Edward Steidle, Washington, D. C.

Proposed by Van H. Manning, George S. Rice, F. G. Cottrell.

Born 1887, Williamsport, Pa. 1907, Williamsport High School, College Preparatory Course. 1911, Pennsylvania State College, B. S. Min. Engrg. 1914, Pennsylvania State College, Advanced Degree E. M. 1908-10, Miner and Mine Sampler, Nipissing Mines Co., Cobalt, Canada. 1911-12, Miner and Millman, Socorro Min. Co., Mogollon, N. Mex. 1912-13, Foreman Miner in charge Mine Rescue Car 8, Lake Superior District. 1913-14, Junior Mining Engineer in charge Mine Rescue Car 6, Western Pennsylvania and West Virginia Districts. 1914-15, Junior Mining Engineer in charge Mine Rescue Car 5, Northwest and Pacific Coast Districts. 1915-16, Asst. Mining Engineer in charge of "The Mine" and Bureau of Mines exhibits, Panama Pacific International Exposition.

Present position: Technical Asst. to Chief Mining Engineer, U. S. Bureau of Mines.

Norman Ayleworth Stockett, Paragon, Ida.

Proposed by C. H. Bowman, Lewis Stockett, Walter H. Aldridge.

Born 1887, St. Louis, Mo. 1904, Grad., High School, Great Falls, Mont. 1909, Montana State School of Mines, Butte, Mont., E. M. 1914, Member, Canadian Institute of Mining Engineers. 1904, Rodman, U. S. G. S. Sun River Irrigation Survey. 1905, Sampling Mill and Laboratory, Trail Smelter, B. C. 1909, Miner, High Ore Mine, Butte. 1910-11, Draftsman, Link Belt Mach. Co. 1912-13, Supt., Willing Min. Co., Gowganda, Ont. 1913-14, drafting and surveying, Centre Star Mine, Rossland, B. C.

Present position: 1915 to date; Supt., Paragon Cons. Min. Co.

Charles Booths Strachan, Mascot, Tenn.

Proposed by J. N. Houser, M. H. Newman, Robert Ammon.

Born 1874, Paisley, Ont., Canada. High School. 1894-1900, Steam Engr. and Stamp Mill, Regina Gold Mining Co., Lake of the Woods, Canada. 1900-03, Tableman and Foreman, Smuggler Mining Co., Aspen, Col. 1903-06, Repair man and Foreman, Cananea Cons. Copper Co., Cananea, Son., Mex. 1906-08, Mill Supt., El Tigre Mining Co., Ysabal, Son., Mex. 1909, Worked for myself. 1910-11, Cananea Cons. Copper Co. 1911-13, Supt., Concentrator, Calumet and Sonora Min. Co., Cananea. 1914, Supt., Bedding Plant, Arizona Copper Co., Clifton and Morenci, Ariz.

Present position: Supt., Concentrator, American Zinc Co.

Charles Edward Straub, Tulsa, Okla.

Proposed by M. M. Valerius, V. H. McNutt, E. G. Woodruff.

Born 1891, Ripley, O. 1911-13, College Mines, Kentucky State Univ. 1913-15, Geol. Dept., Univ. of Chicago, B. S. 1915, Asst. Geol., Kentucky Geological Survey. 1915-16, Asst. Geol., Producers Oil Co.

Present position: Geol., Producers Oil Co.

Lloyd G. Street, Depue, Ill.

Proposed by C. S. Brooks, George C. Stone, J. H. Janeway.

Born 1887, Chicago, Ill. 1909-10, Colorado School of Mines. 1911, Asst. Mine Supt., Cons. Arizona Smelt. Co. 1911, Construction, Ray Cons. Copper Min. & Smelt. Co. 1912, Construction, Los Angeles. 1913, Construction, St. Paul R. R. 1914, Construction for City of Chicago.

Present position: 1915 to date; Engr., Mineral Point Zinc Co.

Thomas Andrew Stroup, Copperhill, Tenn.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe, H. A. Buehler, C. A. Burdick.

Born 1885, Lewistown, Mo. 1903, Grammar Schools, Lewistown, Mo. 1907, Manual Training High School, Quincy, Ill. 1912, Univ. of Missouri, B. S. 1912, Designer on ore dressing machinery and assistant in testing plant, American Concentrator Co., Joplin, Mo. 1913, Designer on coal washing apparatus and plants, Link Belt Co., Chicago. 1914, Engineering and Sales Departments, The Jeffrey Manufacturing Co., Columbus, O.

Present position: Engr. Dept., Tennessee Copper Co.

F. Alfredo Sundt, Corocoro, Bolivia, S. A.

Proposed by William Braden, George E. Montandon, L. D. Ricketts, B. B. Thayer.

Born 1883, Chile, S. A. 1907, Titulado ingeniero de Minas de la Universidad de Chile, Santiago. 1909, Titulado Professor Extraordinario de Metalurgia de la Universidad de Chile, Santiago. 1907, Administrador Mina Elisa de Bordoas, a silver mine in Copiapo, Chile. 1908, First chemist in the Smelter of Caldera (Copiapo) of Sociedad Industrial de Atacama. 1909, Amalgamating and cyaniding tailing dumps of silver ores in Pabellon, Copiapo. 1910, Administrador "Compania Minera Poderosa de Chuquicamata," Antofagasta.

Present position: 1911 to date; Jerente, Compania Corocoro de Bolivia, Corocoro, Bolivia, S. A.

George Monson Taylor, Colorado Springs, Col.

Proposed by Fred. H. Bostwick, J. M. McClave, C. L. Colburn.

Born 1865, Pennsylvania. 1883, General High School. 1887, Univ. of Penn. 1893-95, Gold and Globe Mill, Cripple Creek. 1896-97, Penn-Bullion Mine and Mill, Lewiston, Wyo. 1898, Alaska. 1899, Idaho. 1900-02, Wyoming.

Present position: Genl. Mgr., Milling Dept., Portland Gold Min. Co.

Arthur Davis Terrell, Chicago, Ill.

Proposed by G. S. Brooks, George C. Stone, J. H. Janeway.

Born 1877, Holden, Mo. 1894-99, Missouri School of Mines, B. S. in Civ. Engrg. and Min. Engrg. 1899-1900, Survey and Assaying, Joplin, Mo. and Galena, Kans. 1900-02, Chem., George E. Nicholson Smelter, Iola, Kans. 1902-08, Prime Western Spelter Co., Iola, Kans. 1908-12, Mineral Point Zinc Co., Depue, Ill. 1912-15, Genl. Supt., Prime Western Spelter Co., Iola, Kans., and Tulsa Fuel and Mfg. Co., Iola, Kans.

Present position: Genl. Mgr., Mineral Point Zinc Co.

W. Parsons Todd, New York, N. Y.

Proposed by F. W. Paine, R. H. Gross, J. R. Stanton.

Born 1877, New York. 1901 to date; Quincy Min. Co.

Present position: Vice-Pres., Quincy Min. Co. and Mgr., Copper Sales, Copper Range Co.

Truman Stephen Woodward, Youngstown, O.

Proposed by Thomas McDonald, G. A. Reinhardt, Carl F. Koehler.

Born 1877, Beda, Ky. 1899-1902, Central Univ. of Kentucky, A. B. 1910-11, Harvard Univ. 1902-08, Instructor of Chemistry, Princeton Univ. 1908-10, Prof. of Chemistry, Imperial Pei-Yang Univ., Tientsin.

Present position: 1912 to date; Chief Chemist, Ohio Works, Carnegie Steel Co.

David Cowden Wray, Depue, Ill.

Proposed by G. S. Brooks, George C. Stone, J. H. Janeway.

Born 1875, Winnebago, Ill. 1898, Univ. of Ill., B. S. 1898-1901, Top. Engr., U. S. G. S. 1901-06, Asst. Engr., White Breast Fuel Co., Chicago.

Present position: Asst. Supt., Mineral Point Zinc Co.

Ralph Benjamin Yerxa, Miami, Ariz.

Proposed by B. Britton Gottsberger, F. W. MacLennan, Fred W. Solomon.

Born 1881, Cambridge, Mass. 1903, Mass. Institute of Technology, S. B. 1904, Asst. to Prof. H. O. Hofman, Mass. Institute of Technology. 1905-06, Ferro Machine and Foundry Co., Cleveland, O. 1907, Ricketts & Banks, New York. 1908 to date; Miami Copper Co.

Present position: Asst. Supt., Concentration, Miami Copper Co.

Earl Burdett Young, Butte, Mont.

Proposed by M. H. Gidel, C. W. Goodale, F. A. Linforth.

Born 1882, Ackley, Ia. 1897-1901, Ackley High School. 1901-05, Coe College, Cedar Rapids, Ia., Ph. B. 1908-10, Univ. of Wisconsin, M. A. 1905-08, Teacher of Mathematics and Science in High Schools of Iowa. 1908-10, Instr. on half time in Physics, Univ. of Wis. 1910-13, Instr. and Asst. Prof., Mathematics and Mechanics, Montana State School of Mines, Butte, Mont.

Present position: Asst. in Geological Dept., Anaconda Copper Mining Co.

Associate Member

Charles H. Robinson, Minneapolis, Minn.

Proposed by Horace V. Winchell, George H. Warren, Frank M. Warren.

Born 1866, Minneapolis, Minn.

Present position: For many years owner of mines on Mesabi Iron Range.

Junior Members

Barclay Gladstone Anderson, New York, N. Y.

Proposed by R. M. Raymond, William Campbell, H. L. Carr.

Born 1891, White Sulphur Springs, Mont. 1911-13, Univ. of Cal.

Present position: 1913 to date; Student, Columbia Univ. School of Mines.

Howard Birkett, New York, N. Y.

Proposed by H. L. Carr, William Campbell, E. J. Hall.

Born 1893, Penn Yan, N. Y. 1912, Mercersburg Academy.

Present position: Student, Columbia Univ. School of Mines.

Carl Albert Blaurock, Denver, Col.

Proposed by F. W. Traphagen, William B. Phillips, W. G. Haldane.

Born 1894, Denver, Col. 1900-12, Grade and High School, Denver. With H. Blaurock, Gold and Silver Refiner.

Present position: 1912 to date; Senior Student, Colorado School of Mines.

August Hubert Chatin, Golden, Col.

Proposed by F. W. Traphagen, William B. Phillips, W. G. Haldane.

Born 1894, Walsenburg, Col. 1908-12, Walsenburg High School. 1912, summer, Laborer, Robinson Mine, C. F. & I. Co. 1914, summer, Laborer, Ranch. 1915, summer, Surveyor and Draftsman, S. M. Thompson, Walsenburg, Col.

Present position: 1912 to date; Senior Student, Colorado School of Mines.

Kenneth Sears Ferguson, Golden, Col.

Proposed by J. C. Roberts, William B. Phillips, Harry J. Wolf.

Born 1892, Russel, Kans. 1912, Grad., Manual Training High School, Denver, Col.

Present position: Student, Colorado School of Mines.

Fritz Carl Floss, Corvallis, Ore.

Proposed by H. M. Parks, A. M. Swartley, G. E. Goodspeed, Jr.

Born 1895, Corbett, Ore. 1902-09, Preliminary education in Willsburg Oregon Public Schools. 1909-11, Milwaukie High School, Milwaukie, Ore.

Present position: 1911 to date; Student, Oregon Agricultural College.

Shih-hung Hu, Golden, Col.

Proposed by F. W. Traphagen, Harry J. Wolf, William R. Chedsey.

Born 1895, Kiukiang, China. 1908-13, General education in William Nast College, Kiukiang, China. 1911-13, General education in Tsinghua College, Peking, China.

Present position: 1913 to date; Student, Colorado School of Mines.

Karl F. Klein, New York, N. Y.

Proposed by R. M. Raymond, William Campbell, H. L. Carr.

Born 1891, New York.

Present position: 1911 to date; Student, Columbia School of Mines.

Joseph T. Kupferstein, Brooklyn, N. Y.

Proposed by Robert Peele, R. M. Raymond, H. L. Carr.

Born 1892, New York, N. Y. 1893-1905, New York Public Schools. 1905-10, DeWitt Clinton High Schools. 1910-13, Columbia Univ. School of Mines. 1913-14, Representative, Cyanide Plant and Sampler, Crown Reserve Min. Co., Cobalt, Ont. 1913-14, Caribou, Cobalt (formerly Drummond Representative for this Co. at Dominion Reduction Co., Cobalt, Ont.) 1915, Machine Helper, Canadian Copper Co., Creighton, Ont.

Present position: 1914 to date; Student, Columbia Univ. School of Mines.

Milton Mortimer Levy, Golden, Col.

Proposed by Harry J. Wolf, William B. Phillips, F. W. Traphagen.

Born 1895, Denver, Col. 1907-11, Kearney Military Academy, Kearney, Neb.

Present position: 1911 to date; Senior, Colorado School of Mines.

Colwell Arba Pierce, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1886, Pawtucket, R. I. 1901-04, Manuel Training High School, Kansas City, Mo. 1907-09, Missouri School of Mines and Metallurgy. 1910-12, General Mining Work, Cripple Creek District. 1912-15, Supt., Ruby Copper Co., Patagonia, Ariz.

Present position: 1915 to date; Student, Missouri School of Mines and Metallurgy.

Dudley E. Roberts, Briarfield, S. Bethlehem, Pa.

Proposed by Howard Eckfeldt, Joseph W. Richards, Benjamin L. Miller.

Born 1892, Omaha, Neb. 1907-11, Stamford High School, Stamford, Conn. 1911-13, Lehigh Univ. 1913-14, Time Keeper and Scaler, Greenwood Lumber Co.

Present position: 1914 to date; Senior, Lehigh Univ.

Robert Droege Schmidt, Lakewood, O.

Proposed by Charles H. Fulton, J. Burns Read, Zay Jeffries.

Born 1894, Cleveland, O. 1908-10, Lincoln High School. 1910-12, Lakewood High School.

Present position: 1912 to date; Student, Case School of Applied Science.

William Herzig Stickney, Reno, Nev.

Proposed by F. C. Lincoln, J. O. Greenan, Walter E. Trent.

Born 1886, New York, N. Y. 1910-11, Copper Queen Mining Co., Douglas, Ariz. 1911-12, Amer. Smelt. & Ref. Co., Goldroad, Ariz. 1911-13, Nevada Hills Min. Co., Fairview, Nev. 1914, summer, Nevada Hills Min. Co., Fairview, Nev. 1915, summer, Aurora Cons. Mining Co., Aurora, Nev.

Present position: 1913 to date; Student, Univ. of Nevada.

Harry Allen Sutton, Corvallis, Ore.

Proposed by H. M. Parks, A. M. Swartley, G. E. Goodspeed, Jr.

Born 1895, Geneva, Neb. 1902-09, Preliminary education in Aumsville, Oregon, Public Schools.

Present position: 1909 to date; Senior Student, Oregon Agricultural College.

Frank J. Wiebelt, Arvada, Colo.

Proposed by Harry J. Wolf, William B. Phillips, F. W. Traphagen.

Born 1895, Cleveland, O. 1908-12, Arvada High School. 1914-15, summers, Yak M. M. & T. Co., Leadville, Col.

Present position: 1912 to date; Senior Student, Colorado School of Mines.

Felix Edgar Wormser, New York, N. Y.

Proposed by Robert M. Raymond, William Campbell, H. L. Carr.

Born 1894, Santa Barbara, Cal.

Present position: 1912 to date; Student, Columbia School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Feb. 10, 1916 to Mar. 10, 1916.

This list, together with the list published in *Bulletin* No. 111, March, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916, and brings it up to the date of Mar. 10, 1916.

ADAMS, HUNTINGTON.....	Antofagasta, Chile, So. Amer.
ALBERTSON, MAURICE MERTON.....	Box 744, Cobalt, Ont., Canada.
BARAGWANATH, JOHN G.....	Casilla 140, Lima, Peru, So. Amer.
BARKER, H. A., Care Srs. Stafford & Co., Apartado Correos No. 1,	
	Arequipa, Peru, So. Amer.
BECK, EDWIN L.....	Lovelock, Nev.
BELL, ALAN D.....	Box 592, Baxter Springs, Kan.
BEROLZHEIMER, D. D., Librarian, The Barrett Co., 17 Battery Place, New York, N. Y.	
BLAIR, GEORGE SHEPPARD.....	Research Corp., 63 Wall St., New York, N. Y.
BOISE, CHARLES W.....	Hurley, N. M.
BORDEAUX, ALBERT F. J., Mgr., La Vauvelle Prophyritic Quarry, Corbogne,	
	Nievre, France.
BROWNE, SPENCER C.....	14 Wall Street, New York, N. Y.
BROOKS, JOHN M., JR.....	749 N. Fifth Ave., Knoxville, Tenn.
BUCHANAN, J. R., Genl. Mgr., Imperial Reduction Co., Ogilby, Imperial Co., Cal.	
BURCHAM, WILLIAM D.....	Terlingua, Brewster Co., Tex.
BURNS, W. T., Supt., Electrolytic Plants, Anaconda Copper Min. Co.,	
	Great Falls, Mont.
BURT, CHARLES S.....	1322 Hill Street, Ann Arbor, Mich.
BUTLER, M. C.....	Carbon Coal & Clay Co., Bayne, Wash.
CAHEN, JAMES P., Jr.....	Ray Cons. Copper Co., Ray, Ariz.
CARNAHAN, J. S., Supt., Mexican Lead Co., Apartado 115, Monterrey, N. L., Mexico.	
CARR, HENRY C.....	Care Catlin & Powell Co., 15 Broad St., New York, N. Y.
CARROLL, GEORGE A.....	Care Tropico Mine, Rosemond, Cal.
CHADWICK, JOHN P.....	North Waterboro, Me.
COCHRAN, R. S.....	Y. M. C. A., Youngstown, O.
CONNER, ELI T.....	Union National Bank Bldg., Scranton, Pa.
CRAMPTON, FRANK A.....	Mgr., White Pine Min. Co., Santa Monica, Cal.
CURRIE, DAVID.....	30-31 St. Swithin's Lane, London, E. C., England.
DEVEREUX, W. B., Jr.....	Wilkins and Devereux, 120 Broadway, New York, N. Y.
DICKERMAN, NELSON.....	901 Insurance Exchange Bldg., San Francisco, Cal.
DIETRICH, W. F.....	Humbolt, Yavapai Co., Ariz.
DONNELLY, THOMAS F.....	Care Old Yuma Mine, Box 1026, Tucson, Ariz.
DUNN, THEODORE S., Care Associated Companies,	
	2407 First National Bank Bldg., Pittsburgh, Pa.
EARLE, THEODORE.....	67 Greenann Ave., Hartsdale, N. Y.
ERDLETS, J. F. B., JR.....	45 Broadway, New York, N. Y.
EVANS, MILTON R.....	Mine Inspector, Associated Companies, Plymouth, Pa.
FERRIS, ALBERT LARUE, Chief Engr., Shannon Mine, Arizona Copper Co.,	
	Metcalf, Ariz.
FINK, WILLIAM N.....	Room 423 First Nat'l Bank Bldg., El Paso, Tex.
FISHER, THOMAS E.....	Rua Serpa Pinto 91, Vizen, Portugal.
FRANKE, EMIL A.....	Minocqua, Wis.
FULTON, CHESTER A.....	Apartado 238, Pinar del Rio, P. del R., Cuba.
GALLOWAY, A. D. R.....	Care Millers, Ltd., Komfrodua, West Africa.
GAMBRILL, G. T., Jr., Supt., Shady Side Factory, The Barrett Co.,	
	Hudson Heights P. O., N. J.
GIBBONS, CHARLES A.....	Andes Exploration Co., Chanaral, Chile.
GIFFORD, STANLEY D.....	15 Broad St., New York, N. Y.
GIOLITTI, FEDERICO.....	R. Politecnico, via dell'Ospedale, Torino, Italy.
GRANT, LESTER E., Min. Engr., Braden Copper Co., Rancagua, Chile, So. Amer.	
HACKETT, W. H.....	Muzquiz, Coah., Mex.
HACKFORD, J. E.....	47 Parliament St., Westminster, S. W., London, Eng.
HALLORAN, WILL.....	1011 Kearns Bldg., Salt Lake City, Utah.
HAMMOND, JOHN HAYS.....	120 Broadway, New York, N. Y.
HARLEY, G. TOWNSEND.....	Min. Dept., Detroit Copper Min. Co., Morenci, Ariz.
HASEGAWA, T.....	118 Rokuchome, Aoyama Minamimachi, Akasakaku, Tokyo, Japan.
HENRY, PHILIP W.....	Room 112, 55 Wall St., New York, N. Y.
HENRY, WILBUR E.....	The New Jersey Zinc Co., 55 Wall St., New York, N. Y.
HERON, CHARLES M.....	345 So. Serrano Ave., Los Angeles, Cal.

HERZIG, C. S.	27 William St., New York, N. Y.
HESS, RUSH M.	325 Franklin Place, Plainfield, N. J.
HINCKLEY, E. R.	Santa Rita, N. Mex.
HOFFMAN, LLOYD	135 Howe St., New Haven, Conn.
HOLLAND, L. F. S.	Box 386, Tucson, Ariz.
HOLME, PERCY E.	El Oro Min. & Ry. Co., Ltd., El Oro, Mex., Mex.
HOWARD, L. OGILVIE	Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
IMHOFF, ALEXANDER	1275 Court St., Los Angeles, Cal.
INGERSOLL, J. CURTIS	Georgetown, Cal.
JAMES, C. H.	Kingman, Ariz.
JARMAN, ARTHUR, Asst. Genl. Mgr., Waihi Grand Junction Gold Co., Ltd.,	Waihi, N. Z.
JARVIS, ROYAL P.	Care Cananea Cons. Copper Co., Cananea, Son., Mex.
JENKS, THOMAS H., Cons. Min. Engr.; Supt., Times Min. Co.	Oatman, Ariz.
JONES, E. HORTON	Chief Engr., Canadian Copper Co., Copper Cliff, Ont., Canada.
KEENEY, ROBERT M., Care Snyder Electric Furnace Co., 53	West Jackson Blvd., Chicago, Ill.
KENNEDY, EUGENE P.	720 Foxcroft Bldg., San Francisco, Cal.
KILLIAN, RICHARD B. T., Hardinge Conical Mill Co., 120 Broadway, New York, N. Y.	
KOEHLE, CARL F.	Youngstown Sheet & Tube Co., Youngstown, O.
KONG, S. T.	49 Pokfoolum Road, Hong Kong, China.
LADD, DAVID H.	615 Market St., Sandusky, O.
LAKE, MACK CLAYTON, Care M. A. Hanna Co., 705 Fidelity Bldg.,	Duluth, Minn.
LASTER, FREDERICK G.	Care Detroit Club, Detroit, Mich.
LAUDIG, O. O.	Deseronto, Ont., Canada.
LEMMON, DAVID	Maxfield Mine, Sandy, Utah.
LINDEMUTH, L. B.	Pennsylvania Steel Co., Steelton, Pa.
LINN, W. J.	2629 Michigan Ave., Chicago, Ill.
LINTON, R. A.	Care Army & Navy Club, 107 W. 43d St., New York, N. Y.
LINTON, ROBERT, Min. Engr.; Genl. Mgr., Sierra Cons. Mines Co.,	Hollingsworth Bldg., Los Angeles, Cal.
LOEB, CARL M., Vice-Pres., American Metal Co., Ltd., 61 Broadway, New York, N. Y.	
LOERPABEL, W. HARRISON	Bachelor House, Palmerton, Pa.
LONDON, CLARENCE J.	1525 Chestnut Street, Philadelphia, Pa.
MCALLISTER, J. E., 601 Standard Bank Bldg., 15 King Street, West Toronto,	Ont., Canada.
MCCASKELL, J. A.	321 Felt Bldg., Salt Lake City, Utah.
MCINTOSH, F. K.	Gold Road, Ariz.
McLAUGHLIN, WARNER	Box 482, Cobalt, Ont., Canada.
McMEekin, CHARLES W.	1270 Pine St., San Francisco, Cal.
MACCARTHY, MARION S.	728 First National Bank Bldg., Denver, Colo.
MACCOY, FREDERICK	Care Huntington Adams, Antofagasta, Chile, S. A.
MACKENZIE, K. G., Cons. Chem., The Texas Co., 17 Battery Place, New York, N. Y.	
MALCOLMSON, J. D., Care The Mellon Institute, Univ. of Pittsburgh, Pittsburgh, Pa.	
MARRIOTT, F. A.	Golden Kopje Mine, Lomagundi, Rhodesia, S. Africa.
MASTERS, H. K.	1806 North Ave., Bridgeport, Conn.
MAYNARD, T. POOLE	1321-A-Hurt Bldg., Atlanta, Ga.
MAXON, W. L.	Box 777, Anaconda, Mont.
MEANS, ALAN H.	American Smelting & Ref. Co., Tucson, Ariz.
MEANS, JOHN H.	Room 3533, Equitable Bldg., 120 Broadway, New York, N. Y.
MENTZEL, CHARLES	1935 79th St., Brooklyn, N. Y.
MILLER, ARTHUR J.	1102 Newhouse Bldg., Salt Lake City, Utah.
MILLWARD, WILLIAM	National Roll and Foundry Co., Avonmore, Pa.
MISHLER, RALPH T.	Care The Tigre Mining Co., Esquedra, Son., Mexico.
NEWELL, G. S.	Newell Place, Ave. A, San Antonio, Tex.
PALMER, W. F.	Care Inspiration Cons. Copper Co., Miami, Ariz.
PARRISH, S. F.	Battle Mountain, Nev.
PAYMAL, GEORGE W.	266 Arcadia St., Pasadena, Cal.
PAYNE, ARTHUR C.	608 Vernon Road, Mt. Airy, Philadelphia, Pa.
PENTLAND, WALTER J.	Hotel Cosmopolita, Guadalajara, Jal., Mex.
PERRY, R. P., Genl. Mfg. Mgr., The Barrett Co., 17 Battery Place, New York, N. Y.	
PRATT, M. E.	Ayabaca, Peru, via Paite.
PRATT, WALLACE E.	Box 75, Houston, Tex.
PRICE, H. B.	Abangarez Gold Fields of Costa Rica, Abangarez, Costa Rica.
PROUTY, R. W.	Morenci, Ariz.

- READING, RICHARD W., Cinnamon Bippo Mine, via Abosso, Gold Coast Colony,
West Africa.
- REECE, P. P. Knoxville, Ia.
- REMICK, WALTER L. Virginia Smelting Co., Norfolk, Va.
- RHOADS, ALBERT E. Central Y. M. C. A., Buffalo, N. Y.
- ROBERTSON, JASPER T. Care Mrs. Jentes, 230 W. 99th St., New York, N. Y.
- ROBBINS, P. A., Genl. Mgr., Canadian Mining and Finance Co.,
Ltd., Timmins, Ont., Canada.
- ROESLER, MAX. 235 Lawrence St., New Haven, Conn.
- SAWYER, ARTHUR H. Anniston Ordnance Co., Anniston, Ala.
- SAWYER, HAVEN. 64 Forest Ave., Bangor, Me.
- SCHMIDT, HENRY C. Calle de Hidalgo 117, Monterrey, N. L., Mexico.
- SCHRAPPE, PAUL C., So. Amer. Dev. Co., Apartado 655, Guayaquil, Ecuador, So. Amer.
- SEARS, MORTIMER A. 323 New P. O. Bldg., Denver, Col.
- SEIDEL, VICTOR B. Care Cuba Copper Co., Santiago de Cuba, Cuba.
- SELBIE, CHARLES C. Care Colorado Mining Co., Aroroy, Masbate, P. I.
- SEMPLE, R. A. 23, The Lochiel, 302 Buffalo Ave., Niagara Falls, N. Y.
- SHANKS, D. W. Box 377, Nevada City, Nevada Co., Cal.
- SHERA, EDWARD L. Apartado 91, Mazatlan, Sinaloa, Mex.
- SMITH, HARVEY E., Care The Associated Companies,
2407 First National Bank Bldg., Pittsburgh, Pa.
- SMITHER, THOMAS M. Box 828, Reno, Nev.
- SOPER, ELLIS, Chief Engr. of Design & Constr., Cuban Portland Cement Co.,
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Date of Election.	Name.	Date of Decease.
1886	*Allison, Robert	Feb. 3, 1916.
1871	*Harden, J. H.	Feb. 14, 1916.
1891	*Law, John B.	Jan. 17, 1916.
1915	*Payne, James Harvey	Jan. 28, 1916.
1877	*Richards, Henry	Mar. —, 1915.
1874	Stafford, C. Edward	Nov. —, 1915.
1914	*Watson, Ernest E.	Oct. 19, 1915.

*Member.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Water Problem at the Old Dominion Mine

BY P. G. BECKETT* GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

THE problem of handling the large quantities of water encountered in the Old Dominion mine presents many features of interest. In the present paper are discussed the probable sources of water, the pumping equipment, the increased flow into the mine in the early part of 1915, culminating in the flooding of the 18th and 16th levels in March of that year, and the difficulties that were met with during the high-water period.

I. PRELIMINARY

The Old Dominion mine, situated 1 mile north of the town of Globe, Ariz., has long been one of the prominent copper producers of the State, and until the opening up of the copper schist deposits in the neighboring mines of Miami, was responsible for the greater part of the copper output from the Globe district.

As the gradual exhaustion of its orebodies near the surface led to the downward development of the mine, more water was encountered in the western portions directly underlying the dacite formation until the drainage problem became a serious factor in the operation of the mine and in the economical extraction of the ore.

A heavy inrush of water on the 10th level of the mine in 1906, preceded by still earlier water difficulties, rendered imperative the immediate installation of a pumping equipment capable of handling large flows of water. While the Old Dominion has been recognized as a wet mine since its early days and one in which the complete control of the water situation was of paramount importance in the development of the lower levels, it is chiefly in the last nine years, since the installation of the increased pumping equipment, that the mine has developed and handled large quantities of water.

Its maximum pumping record was reached in the first six months of 1915, during which time, following a prolonged period of abnormal precipitation, the pumps handled 1,624,740,000 gal. of water; while in the

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month of March alone, over 407,000,000 gal. of water were pumped from the mine.

II. PROBLEM SOURCES OF WATER

For a proper understanding of the fundamental conditions affecting the flow of water into the mine, a brief description is necessary of the topography and structural geology of the surrounding district, together with a review of the prevailing climatic conditions and a detailed study of the drainage of the Old Dominion area.

TOPOGRAPHY AND STRUCTURAL GEOLOGY

Globe is situated at an elevation of 3,500 ft. above sea level on Pinal Creek, the main drainage channel of the district, which rises in Pinal Mountains and flows on a very even grade northwest to where it joins the Salt River.

The topography surrounding Globe is irregular, but consists in general of a broad valley of rolling and slightly hilly lowland country, rising on either side of Pinal Creek and stretching to gradual rising foothill country, which in turn to the southwest flanks Pinal Range, the main mountain range of the district (with an elevation of 7,850 ft. at its highest peak), and to the northeast is terminated by the less steep and less conspicuous Apache Mountain range. Almost the entire portion of the valley zone to the west of Pinal Creek and country east of it south of the town of Globe, lies in Gila conglomerate, the most recent gravel formation, while north and east of Pinal Creek, in the immediate vicinity of the Old Dominion and United Globe mines, the surface formation consists of a much faulted and geologically complex area of limestones and quartzites intruded by diabase, mainly represented by a monzonite sill interlocated between the Cambrian "Mescal" limestone and the upper Cambrian or "Troy" quartzite and by several small dikes of quartz monzonite porphyry which also cut the diabase. These formations dip gently to the southwest toward Pinal Creek, and west of the main "A" shaft are overlain by a flow of dacite, which is in turn covered by the more recent Gila conglomerate to the west of Pinal Creek.

In the area northeast of the main "A" shaft, the less easily eroded quartzites largely form the capping of the ridges of the surrounding hills, while the gulches and basins are carved in the comparatively easily weathered diabase, the gulches often following the fault zones.

The lower foothills of the Pinal Range, south and southwest of the mine, change from the conglomerate of the valley or trough zone to the pre-Cambrian Pinal schist of the main range, large areas of which, intruded by granitic rocks, stretch to the northwest of the quadrangle. A columnar section of the geologic formations of the district is shown in Fig. 1.

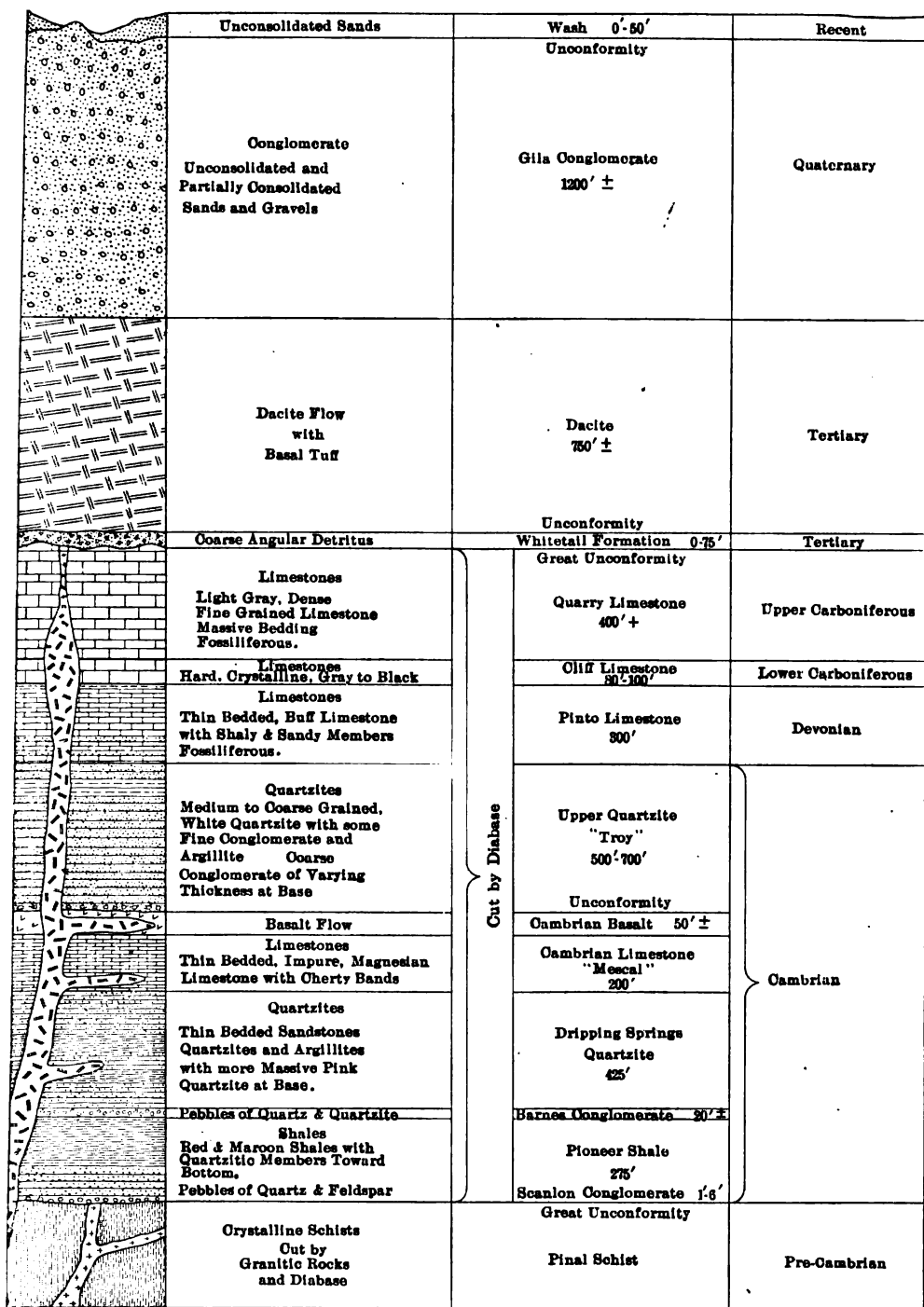


FIG. 1.—COLUMNAR SECTION SHOWING GEOLOGIC FORMATIONS, GLOBE DISTRICT.

The main "A" shaft of the Old Dominion mine is sunk in the foot wall of the Old Dominion vein, a fault zone with a northeast-southwest trend, and from which, occurring in the form of big lenses or shoots, the principal ore extraction has come.

Southwest of the "A" shaft, as has been described before, the sedimentaries dip gently to the west and are covered by a lava flow post-Mesozoic in age, which increases in thickness from its taper point, where it overlaps the older formations near the "A" shaft to some 350 ft. directly below Pinal Creek, thus effectively covering all trace of the ore zone in its westward trend. The Quaternary Gila conglomerate, as now exposed, shows some 2,000 ft. west of the "A" shaft. This formation also gains in thickness toward the west, and reaches a thickness of 1,000 ft. a short distance southwest of Pinal Creek. Between the dacite and upper gravel formation, there occurs a mass of Madera diorite, which was probably thrust-faulted into its present position in post-dacite time. Its thickness directly below Pinal Creek is somewhat over 300 ft. Fig. 2 shows a section of the structural formation from the main shaft west to Pinal Creek, and clearly indicates the relation of these superimposed non-mineralized formations to the underlying orebodies and to the formations favorable for ore exploration.

The Gila gravel formation in character ranges in size from fine sands through coarse gravel and pebbles up to big boulders. The pebbles comprising the conglomerate are more often angular than round, and where this formation has been cut, fragments are disclosed of all the older rocks of the area including schist-granite, diorite, limestone, quartzite and diabase formations. Its bedding is usually fairly well defined. The Madera diorite underlying it, in texture somewhat resembles a decomposed granite. Wherever encountered, it has been highly shattered, and where exposed at the surface is decomposed and crumbly in character, and from very recent investigation, intensely porous. The dacite flow immediately underlying the sedimentaries (omitting a thin bed of gravel formation accumulated over the sedimentaries before the outflowing of the lava) is a tough, comparatively firm rock when dry, and although it is the formation in which the water is always encountered in the western portion of the mine, is not in itself particularly porous, as proved by the not inconsiderable amount of work that has been done in it without encountering water. But once fractures in the dacite allow the downward seepage of water from the diorite above, the formation loses all pretense of firmness, absorbs water like a sponge, and becomes sandy, crumbly, and difficult to hold.

CLIMATIC CONDITIONS

Climatically and physically, the Globe District is typical of the mountain section of Arizona. In an ordinary normal year, the annual pre-

precipitation averages only 17 in. Most of this precipitation in the summer months of July and August is in the form of sudden and violent cloudbursts, which, while bringing immense quantities of water down the main drainage channels, is not of the character of precipitation that sinks very deep into the sun-parched ground. Winter rains, when they do come, are of the heavy, soaking type.

The surface of the ground throughout this area is mostly rocky and destitute of much soil. Hence, the district is largely barren of vegetal and tree growth, and, except on the higher flanks of the mountain ranges, where melting snows and deeper soil prolong tree life, and in special irrigated sections, the district physically is arid in the extreme.

DRAINAGE AREA OF OLD DOMINION MINE

1. General Description of Drainage Area

The particular portion of Pinal Creek drainage area in which the mine is situated does not exceed 40 square miles, and may be considered as stretching from 3 miles south of Globe, where the watershed drains into the San Carlos Basin, a tributary of the Gila River, to the line where the northern limit of the company's property crosses Pinal Creek. The rainfall as registered at Globe for the years 1910 to 1914, and for the first six months of 1915 is shown in Table I. It is important to note, that of

TABLE I.—*Total Rainfall Registered at Globe, Ariz., for Years 1910 to 1915*

		Inches		
		1910.....	10.72	
		1911.....	21.36	
		1912.....	17.27	
		1913.....	14.72	
		1914.....	23.47	
Last 6 Months, 1914				
	Inches			
July.....	4.90			
August.....	3.17			
September.....	0.47			
October.....	3.24			
November.....	1.40			
December.....	5.59			
	18.77			
		First 7 Months, 1915		
			Inches	
		January.....	4.52	
		February.....	2.92	
		March.....	1.10	
		April.....	2.15	
		May.....	0.61	
		June.....	0.25	
		July.....	2.70	
			14.25	

the 23.4 in. of precipitation in 1914, 18.77 in. was recorded in the second six months, making a total of 30.3 in. from July 1, 1914, to June 30, 1915, or nearly double the normal precipitation for 12 months. No accurate figures are available on the run-off from the special area under considera-

tion, or indeed, from the entire length of the Pinal Creek and Miami wash drainage area, but it may be taken for granted that it varies considerably with the nature of the rainfall; i.e., whether the latter originates in the form of cloudbursts in the mountains or as a steady downpour over the entire watershed. Also, it is largely dependent on the condition of the ground surface and the creek bed in the drainage area. It was very noticeable during the past winter, when the total monthly precipitation in December, January, and February was not only in the form of steady, soaking rains, but also as periodical heavy floods. The water in Pinal Creek flowed steadily without cessation during the entire winter months, and increased in the spring by the late melting snows in the Pinal Range ran as a good-sized stream, carrying between 20,000,000 and 40,000,000 gal. per day, well into the month of June, an unmistakable criterion of the saturated condition of the surrounding ground.

Well measurements in the district showed that the water level in certain formations rose as much as 40 ft. in three months, the rise being very rapid during the latter period of the heavy rainfall, and apparently taking place after the ground had absorbed all the previous moisture possible. Recent weir measurements, taken at the Old Dominion weir at the south end of its property, showed at the height of a summer flood, when over 2 in. fell in 8 hr., that as much as 300,000 gal. per minute was flowing in the creek.

2. Different Sources of Water in the Mine

Owing to the peculiar geological conditions previously described, the Old Dominion has, in its problem of handling water in the mine, two distinct sources to contend with; one the east-side water, and the other the west-side water.

The east-side water is that coming into the lowest levels of the mine, and is doubtless the result of the lowering, in the vicinity of the mine workings, of the normal ground-water level of the entire district. The east-side water is encountered principally in shale and quartzite formations, and rarely in the denser diabase. The driving of each successive lower level drains the water from the level above. It has a normal temperature of 90° F., or 25° warmer than the west-side water. While this flow is of considerable importance, it is fairly constant in quantity, averaging about 2,000,000 gal. per day and is not subject to the variations of the west-side water. The east-side water source, therefore, will not be discussed in detail in this paper.

The west-side water is the water held above the dacite formation in the west end of the mine, and while this water is also ground water, it is prevented from sinking lower in common with the general drainage of the mine area by the impervious layer of dacite that separates it from the underlying mine workings. As a consequence, although mining opera-

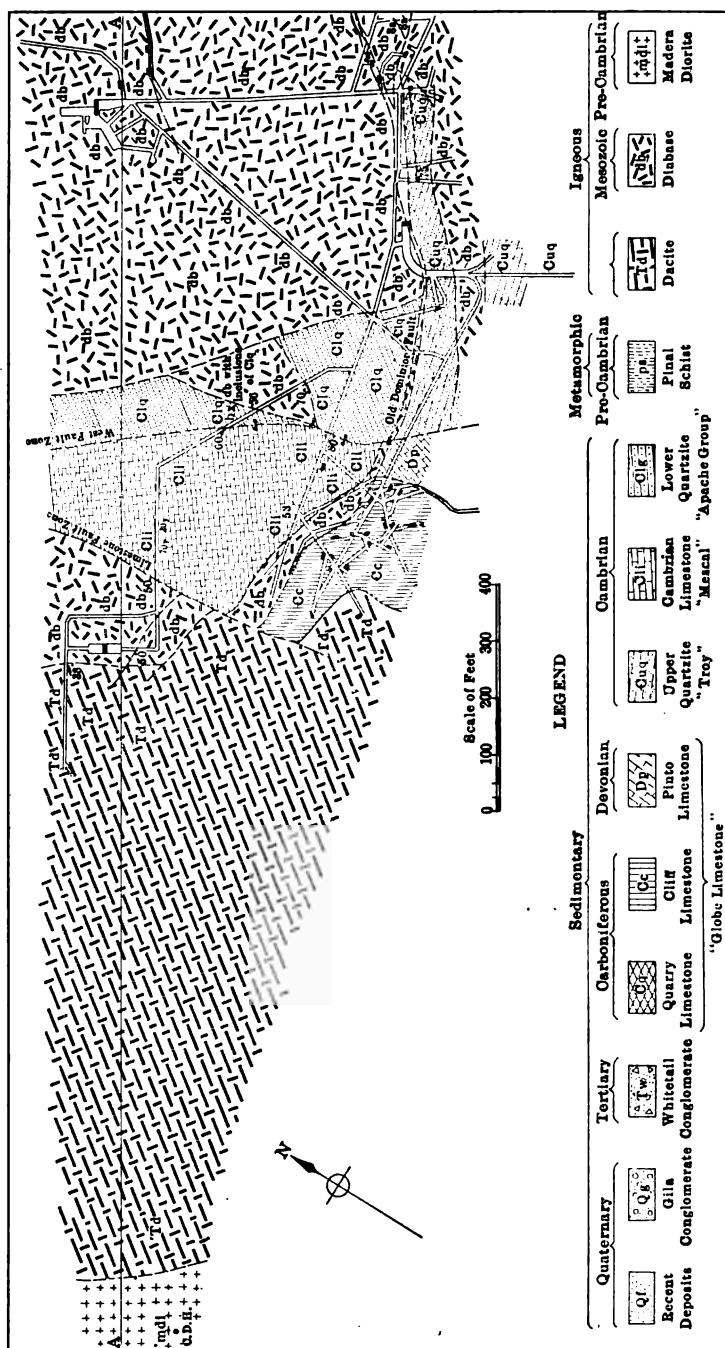
tions are proceeding on the 18th level in the west section of the mine, the lowest drainage point for this west-side water is on the 12th level, 600 ft. above, where the water is tapped through fractures in the dacite. This water contains a relatively small proportion of solids, and is excellent for domestic purposes. Inasmuch as the handling of the west water has been in recent years the most troublesome of the mine's pumping problem, it is this particular source that will be considered in detail.

3. Description of Underground Workings Adjacent to the West Drainage Source

As a matter of recent history, and as leading up to an account of the disastrous inrush of water on the 12th level in March, 1915, it may be stated briefly that as the mining of the orebodies, to their western limits immediately under the dacite capping, proceeded on the upper levels, so did the necessity of draining the water contained in and above the dacite to a continually lower level become more evident. Twice before the latest inrush of water was the safety of the mine imperilled by the sudden and unexpected tapping of large volumes of water held above the dacite, which were encountered while mining ore along the dacite contact.

In 1906, the breaking in of the water above the 10th level drained within a very short time the water on the 8th and 9th levels above, and it was deemed necessary to install adequate pumping facilities on the 12th level and to drain all water to that level in order to extract the ore remaining above that horizon that lay close to the dacite formation. This was accordingly done, and the intake of the water lowered 200 ft. to the 12th level, which in turn, in a general way, but as will be shown later, somewhat imperfectly drained all of the country above. A plan of the 10th level is shown in Fig. 3. In 1914, therefore, the 12th level was the main drainage and pumping level for all the water originating on the west side of the mine, and also the main relay level for all water pumped from the center and eastern portions of the mine below that level. Fig. 4 shows a plan of the pertinent features on the 12th level from the shaft and pump station out to the original draining point. The water drift first run to carry the water back to the pump station is marked as AA, and shows the formation through which it passed before tapping the water well within the dacite. Later, on account of the original drift being directly above an orebody on the 13th level and the seepage of the water from the drift interfering with the stoping operations below, a new drift BB was run well into the foot wall and away from any possible ore below. At the same time, the old drift was dammed up.

During 1913 and 1914 active and continuous efforts had been made to drain what is known as the Zero East, and Zero West section of the mine marked in circles in Fig. 4. Considerable ore remained between the 12th



and 11th levels in this territory, but although a considerable amount of work had been done in the dacite itself in the immediate vicinity of the orebody, water was never encountered in sufficient quantities to drain the country above. But as an outlet for the water that was being produced by this section of country, a drift *CC* cutting across the old water drift was driven to the new water drift *BB*. This drift *CC* is of interest because at approximately the point *Z* the first inrush of water broke through last March. In short, previous to the inrush of new water, the drift at *BB* was delivering all of the water coming from the draining point in the dacite marked *W* in Fig. 4. A comparatively insignificant amount was coming through the drift *CC* from the undrained Zero country. The old water drift *AA* was dammed near its western end, but was opened east of the drift *CC* to opposite 13-1-11 raise marked *X* on the map. This raise was used for the purpose of carrying air from the *C* shaft to the 13th level. Below the 12th level ore was being extracted on the 13th, 14th, 15th, and 16th levels, each 100 ft. apart. Below the 16th level, the main shaft had been sunk to the 18th level, and pumps had been recently installed to handle the water on that level coming from the central and eastern portions of the mine.

GENERAL

In 1909, with the idea of eliminating any possible seepage of water from the creek bed into the mine, a concrete dam was put down to bed-rock in Pinal Creek by the Old Dominion Copper Co. a quarter of a mile south of the mine. All water running in the creek was thus forced over the dam and by means of a flume was kept out of the creek for a distance of 2,300 ft.

Because no difference in the water flow in the mine was apparent when the water was bypassed, this practice was in time discontinued. Recent comprehensive but at present incomplete experiments with weirs and pipe readings of the loss of water through the sands and gravels taken at various points in the creek, also measurements of the level of the standing water in the gravels over the various underlying formations, tend to indicate that perhaps the flume was not carried far enough down the creek to insure the carriage of the water past and over the most absorbent formations. Fig. 5 shows the annual water flow on or above the 12th level of the mine in the year 1913, together with the rainfall curve for those 12 months, and is typical of the recent normal year's water-pumping operations from the west side of the mine. In addition to the flow records on the chart as coming from the 10th and 12th levels, there was pumped from the lower levels of the mine up to the 12th level a fairly steady flow of between 1,500,000 and 2,000,000 gal. per day.

The flow diagram recorded in Fig. 5 seems to indicate that in normal years a small increase of water is recorded in the mine, following with a

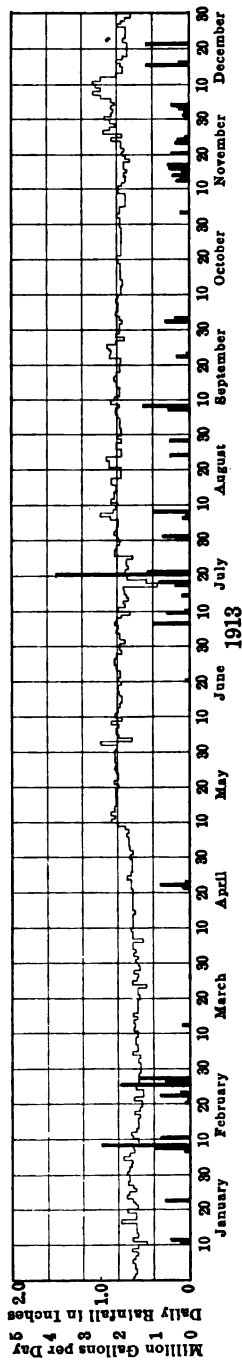


FIG. 5.—WATER FLOW ON AND ABOVE 12TH LEVEL AND RAINFALL IN 1913.

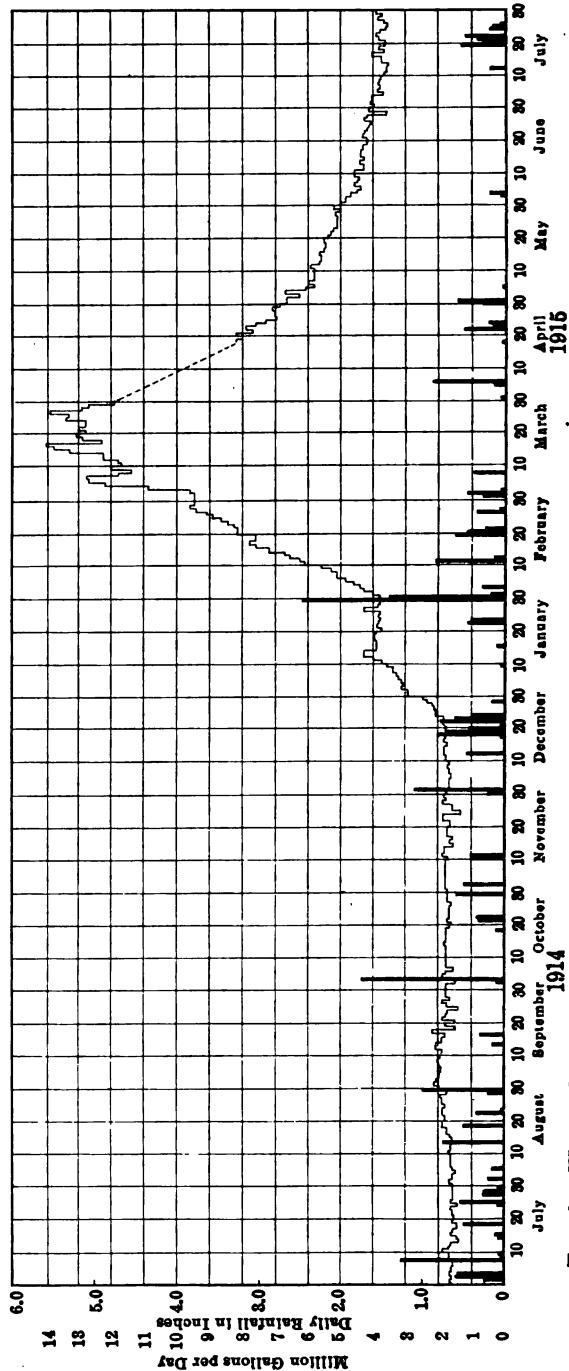


FIG. 6.—WATER PUMPED FROM MINE, JULY 1, 1914, TO JULY 31, 1915, AND RAINFALL DURING SAME PERIOD.

lag of 6 or 7 weeks behind the summer rains. Fig. 6 shows the quantity of water pumped from the mine for the 13 months from July 1, 1914 to July 31, 1915, together with a curve of the rainfall during that period. It indicates how the period from July 1 to Dec. 15 was what may be termed the "saturation" period, and that although the precipitation was considerable during these months, there was no increase in the flow of water from the western portions of the mine. It indicates how the $5\frac{1}{2}$ in. of rain which fell in December, after the ground had absorbed all the moisture possible, was quickly reflected by the sudden rise in the mine flow, but how apparently following on a period from Jan. 1 to Jan. 20 of comparatively little precipitation, the mine water showed a tendency to decrease. The precipitation of Jan. 22 to 24 was, however, immediately felt in the mine, and the additional 5 in. of rain that fell on a few consecutive days at the end of January and beginning of February caused the mine flow to increase steadily until the middle of February. A few days drop at this time when the 10th and 12th level flow amounted to over 7,750,000 gal. and the total amount pumped to over 10,000,000 gal. per day, led us to hope that the worst had been reached. But following a rainfall of $1\frac{1}{2}$ in. about Feb. 20, the water flow increased steadily on the 10th and 12th levels until Mar. 4, when the "break" came in the mine and we handled in excess of 14,000,000 gal. on that day.

III. PUMPING EQUIPMENT

NORMAL EQUIPMENT

1. Preliminary

The normal pumping equipment of the Old Dominion consists of steam pumps on the 10th, 12th, and 14th levels which pump direct to the water storage tanks on surface and to the drain tunnel which runs from a point in the main "A" shaft 235 ft. below the surface discharge level to Pinal Creek.

Electric pumps are installed on the 14th, 16th, and 18th levels, which are all approximately 200 ft. apart, and these pumps lift the water to the main pumping station on the 12th level, whence it is relayed to the surface or to the drain tunnel.

A boiler plant is situated adjacent to the main "A" shaft to generate steam for the steam-operated pumps underground, while electric power for the lower level pumps is transmitted from the central power plant near the smelter. The normal boiler equipment at the "A" shaft consists of five 335-hp. Stirling boilers, equipped with superheaters and economizers, and one 250-hp. Stirling boiler, while to these were added during the flood period one 250-hp. Scotch marine boiler and one 200-hp. boiler of the same type.

During the heavy water period in February, March, and April, compressed air was supplied for the air lifts and sinking pumps from the central power plant.

The "A" shaft, which is the main hoisting shaft for the property, consists of five compartments: Two skip hoisting, two cage hoisting, and one reserved for steam lines, air lines, water columns, and electric cables.

A pump winze runs from the 10th level down to the 18th level, and on the 11th level at the "A" shaft the water columns and main steam lines are transferred from the shaft to the pump winze, and thence down to the main pump station on the 12th level. All water columns from below the 12th discharging to that level are also brought up in this pump winze.

2. Steam Lines

The steam lines from the main boiler plant consist of one main line, 6 in. in diameter down to the 11th level and 5 in. thence to the 12th level; also one auxiliary steam line, 8 in. in diameter to the 10th level, 6 in. from the 10th to the 12th and 4 in. from the 12th to the 14th. A 5-in. line branches from the auxiliary line at the 12th level, and the two 5-in. lines are cross-connected at this point with the main lines down the shaft to allow the 12th level pumps being run off either steam line in case of repairs to the other. At the 10th level a 4-in. steam line is run to each of the triple-expansion pumps from the 8-in. auxiliary line, and at a steam receiver on the 12th level into which the two 5-in. lines connect, 4-in. headers are run to the four triple-expansion flywheel pumps.

The triple-expansion Prescott pump on the 12th takes its steam through a 4-in. line run from the auxiliary 6-in. line.

3. Water Columns

There are three 12-in. and one 14-in. water columns running from the 12th level up the surface, allowing discharge either at surface or at the drain tunnel. The 12-in. columns are of steel pipe with cast-iron flanges; the 14-in. column is wood lined. A fifth water column, 10 in. in diameter, was put in during the high-water period from the 10th level up to the drain tunnel. Below the 12th level two 10-in. water columns run from the 18th level up the pump winze, which columns are cross-connected at the 16th level with the electric pumps on that level. A third 10-in. water column discharges from the electric pump on the 14th level to the 12th pump station. Fig. 7 shows a diagram (not to scale) of the water columns and steam lines and their connections to the various pumps.

The 12th level is 197 ft. below the 10th level; the 10th level is 436

ft. below the drain tunnel, which in turn, is 235 ft. below the point of discharge into the surface water-storage tanks. In other words, the 12th level pumps in discharging to surface, have a lift of 868 ft. and when discharging at the drain tunnel, a lift of 633 ft.

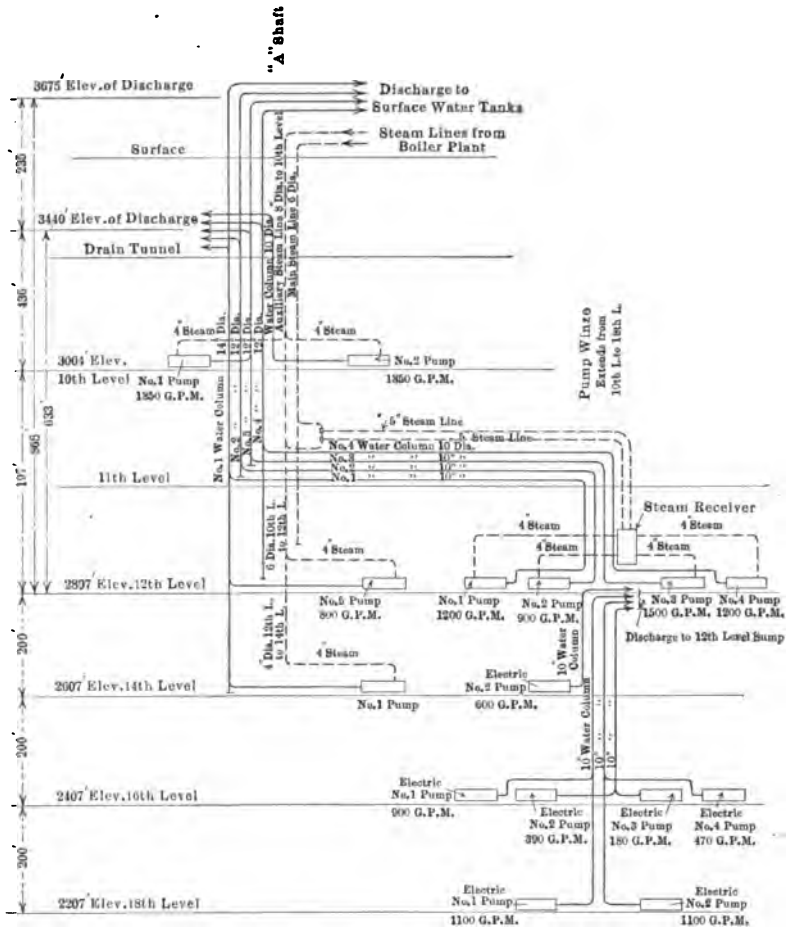


FIG. 7.—DIAGRAM OF WATER COLUMNS AND STEAM LINES WITH CONNECTIONS TO PUMPS.

4. Pumping Machinery and Pump Stations

On the 10th level are two triple-expansion duplex type, 15 by 27 by 39 Prescott pumps, capable at the lift of 436 ft. to the drain tunnel of delivering 1,350 gal. per minute. On the 14th level there is a similar type pump which at the lift of 632 ft. to the drain tunnel is capable of delivering 800 gal. per minute.

On the 12th level there are four triple-expansion flywheel pumps

14-26-26-26 with 36-in. stroke, of Nordberg manufacture, capable at 56 r.p.m. of delivering from 900 to 1,500 gal. per minute, according to the size of the water plungers, which vary in the different pumps from $6\frac{5}{16}$ in. in pump No. 2 up to $8\frac{1}{4}$ in. in pump No. 3.

The triple-expansion Prescott pump on the 12th level is 15 by 27 by 39 with a 24-in. stroke. This pump is used as a reserve and at a speed of 35 r.p.m. can deliver 800 gal. to the drain tunnel.

These steam pumps are run condensing. Steam pressure at the pumps averages about 134 lb. and the steam temperature at the pumps is approximately 350°.

Recent tests run under operating conditions show the steam consumption by the triple-expansion flywheel pumps, which are the ones used in normal times, to be 18.0 to 18.2 lb. per water horsepower-hour.

TABLE II.—*Old Dominion Copper Mining & Smelting Co.*
Mine Pumps

Pump Number	Location, Level	Type of Pump	Steam Cylinders, Inches	Motor Horsepower	Water Plunger, Inches	Stroke, Inches	Speed, Revolutions per Min.	Capacity per Min., Gal.	Lift, Feet	Location of Discharge
1	10th	Triple-expansion duplex	15-27-39		12½	24	30	1,350	436	Drain tunnel
2	10th	Triple-expansion duplex	15-27-39		12½	24	30	1,350	436	Drain tunnel
1	12th	Triple-expansion flywheel	14-26-26-26		7½	36	56	1,200	Drain tunnel 633 Surface 868	Drain tunnel and surface.
2	12th	Triple-expansion flywheel	14-26-26-26		6½	36	56	900		
3	12th	Triple-expansion flywheel	14-26-26-26		8¼	36	53	1,500		
4	12th	Triple-expansion flywheel	14-26-26-26		7¾	36	56	1,200		
5	12th	Triple-expansion duplex	15-27-39		9¼	24	35	800	633	Drain tunnel
1	14th	Triple-expansion duplex	15-27-39		9¼	24	35	800	833	Drain tunnel
2	14th	Vertical triplex, double-reduction		70	12	16	30	600	200	12th level
1	16th	Vertical quintuplex, single-reduction.		150	8	18	51	900	400	12th level
2	16th	Vertical triplex, double-reduction		42.5	10	16	28	390	400	12th level
3	16th	Vertical triplex, double-reduction		40	7	10	42	180	400	12th level
4	16th	Vertical triplex, double-reduction		70	10	16	34	470	400	12th level
1	18th	Horizontal quintuplex, single-reduction.		250	9	18	49	1,100	600	12th level
2	18th	Horizontal quintuplex, single-reduction.		250	9	18	49	1,100	600	12th level

In normal periods, any water on the 10th level is siphoned to the 12th level through the "C" shaft, which makes the latter level the highest pumping point for all water. Of the four flywheel triple-expansion pumps on the 12th level, No. 2 takes its water through a pipe from the concrete dam at the end of the 12th level water drift, and this water is

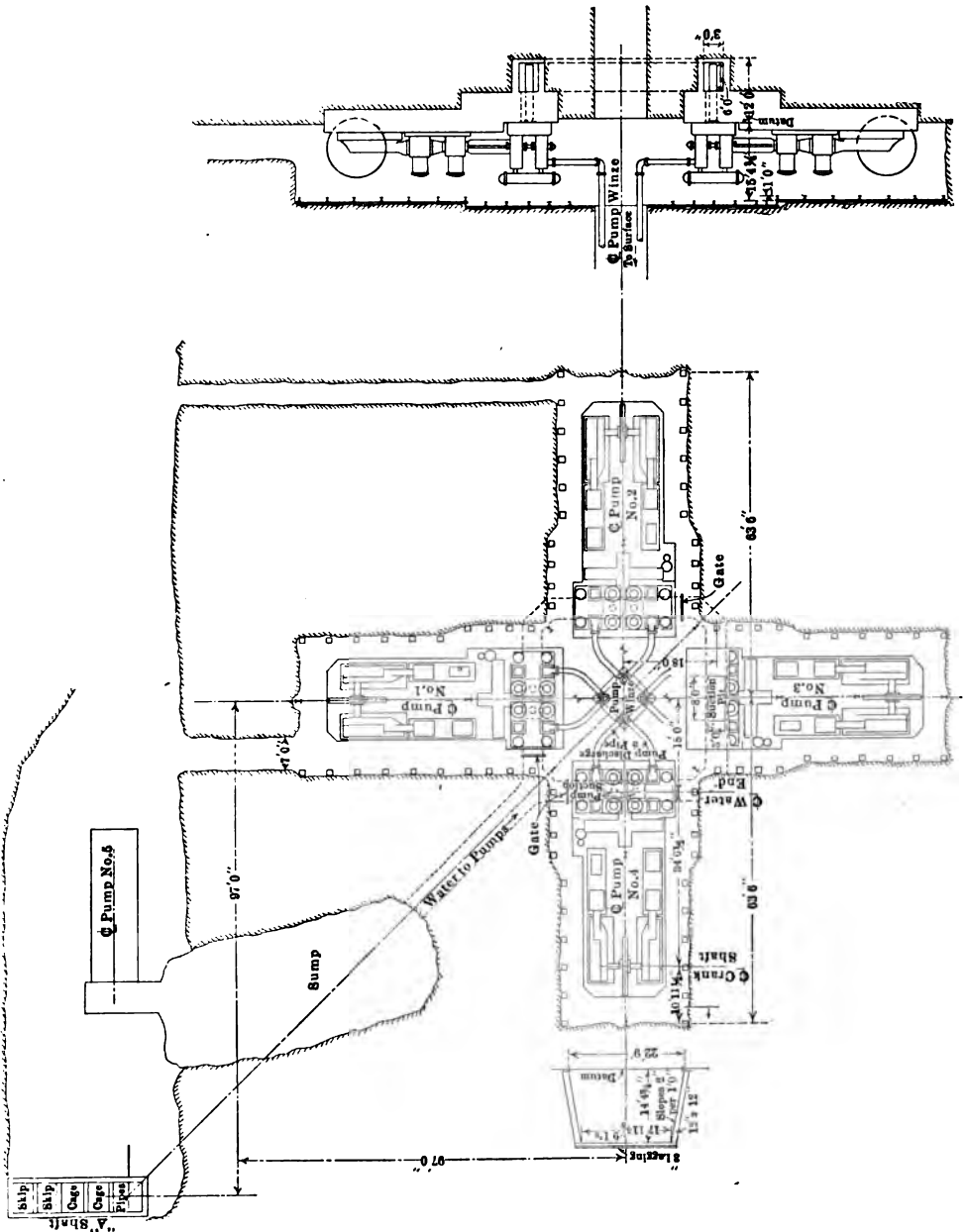


FIG. 8.—ARRANGEMENT OF PUMP STATION ON 12TH LEVEL.

used for domestic and boiler purposes. Nos. 1, 3, and 4, together with the triple-expansion Prescott (when in use), take their water direct from the station sumps, and pump either to surface or to the drain tunnel, according to where the water is needed. The water necessary for the concentrator and fire-line purposes goes direct to surface and the surplus to the drain tunnel. A fixed quantity of this surplus is delivered daily to the Miami Copper Co. while the remainder runs to waste in the creek. Below the 14th level the pumps are all electric driven; on the 16th there is one vertical quintuplex, single-reduction, Aldrich pump with 150 hp. alternating-current motor. This pump lifts at the rate of 900 gal. per minute from the 16th to the 12th level. Housed in the same station with the quintuplex pump are three vertical, triplex, double-reduction pumps, varying in size and capacity, and run by direct current. The capacity of the biggest of these triplex pumps is 470 gal. per minute, and of the smallest, 180 gal. per minute. These three pumps also lift from the 16th to the 12th level. On the 18th level, 200 ft. below the 16th, there are two horizontal, quintuplex, Aldrich pumps, both with 250-hp. alternating-current motors. The capacity of both these pumps is 1,100 gal. per minute from the 18th to the 12th level. In Table II is given a list of the pump equipment which was being used at the end of 1914, and to which, as will be shown later, were added many small emergency pumps and air lifts during the flood period.

The arrangement of the pump station on the 12th level is somewhat unusual and of interest. It is in the form of a maltese cross, as reference to Fig. 8 will show. In the center of the cross is the pump winze which, as has been noted previously, is used for pump columns and steam lines from the 11th to the 18th level. Each arm makes, in a manner, a separate station, and is 63 ft. 6 in. long, measured from the center of pump winze. The width of each arm of the station is 22 ft. 9 in. at the floor line; and the height is 14 ft. 4¾ in. Each of the four arms of this cruciform station houses one of the triple-expansion flywheel pumps with its discharge end set toward the pump winze in the center, and each having its own discharge column. This particular arrangement has considerably simplified the piping in the station, by means of short and easy sweeps into the main water columns. The way in which this station has been laid out is very satisfactory, but doubtless costs more for excavation than a plain rectangular one of the same cubic footage. A sump which holds in the neighborhood of 80,000 gal. is situated between the station and the shaft. The sump discharges the water into a concrete conduit supplying the suction pits of the four pumps. This conduit has doors in it, permitting the shutting down of any one suction pit for cleaning or inspection. The conduit is 12 ft. below the pump station floor line and is 3 ft. wide. The intake is screened to prevent wood chips and waste getting to the valves.

EMERGENCY EQUIPMENT

1. Auxiliary Pumps

Although several small emergency pumps were installed on the 12th and 10th levels toward the end of February when the regular pumping equipment was reaching its limit, these emergency pumps were only used for a comparatively short time, and were later replaced by air lifts on account of the latter's much greater pumping capacity and freedom from shutdowns.

Two 14 by 12 by 8 Prescott sinking pumps were installed on the 10th level toward the end of February. They helped out the two triple-expansion pumps on that level when the latter were reaching their limit, while on the 12th level there was installed a 500-gal. duplex piston pump and one 14 by 12 by 8 Prescott sinker. These emergency pumps were all operated by air.

2. Air Lifts

Early in March an air lift was installed from the 12th to the 10th level and later one from the 10th to the drain tunnel. The submergence on the air lift from the 12th to the 10th level was 177 ft., or 47 per cent. The top 100 ft. of the column was made of 12-in. pipe, the bottom 100 ft. and the two legs were of 10-in. pipe. The air was supplied from a 4-in. line at 80 to 90 gage pressure.

TABLE III.—*Tests of Air Lifts. Old Dominion Copper Mining & Smelting Co.*

Test No.	Net Lift in Feet, Excluding Friction	Submergence in Feet	Submergence, Per Cent.	Gage Pressure of Air in Lb.	Amount of Opening in 4-in. Air Valve	Water Pumped by Air Lift, Gal. per Min.	Free Air Used by Air Lift, Cu. Ft. per Min.	Cu. Ft. of Free Air Used per 1,000 Gal. of Water	Efficiency = Water Hp. at Air Lift to 1 Hp. at Compressor, Per Cent.	Pounds Steam Used at Compressor per Water Horsepower-Hour
A-1	200	177	47.0	80	¼ turn	1,011	1,353	1,338	28.8	70.8
A-2	200	177	47.0	80	½ turn	1,679	1,809	1,080	36.0	50.3
A-3	200	177	47.0	80	1 turn	1,793	2,262	1,261	30.9	58.8
A-4	200	177	47.0	80	1¼ turn	1,925	2,658	1,375	28.4	64.4
A-5	200	177	47.0	80	Full	1,965	3,219	1,638	25.5	71.6
B-1	431	188	30.4	90	1 turn	1,317	3,484	2,645	32.9	57.7
B-2	431	188	30.4	90	2 turns	1,291	3,832	2,968	27.0	64.8
B-3	431	188	30.4	90	3 turns	1,291	3,919	3,035	26.4	66.2
B-4	431	188	30.4	90	Full	1,325	4,089	3,086	26.0	67.2
C-1	431	188	30.4	90	1 turn	1,122	3,051	2,718	29.4	59.5
C-2	431	188	30.4	90	2 turns	1,233	3,306	2,681	29.9	58.4
C-3	431	188	30.4	90	Full	1,233	3,484	2,825	28.4	61.3

Tests in group "A" were on the air lift pumping from 12th level to 10th level. Tests in group "B" and group "C" were on the air lift pumping from 10th level to drain tunnel.

Short tests on this air lift showed that it handled to the 10th station over 1,650 gal. per minute, with an air consumption of 1,080 cu. ft. per thousand gallons. The highest efficiency obtained was 36 per cent.

In the lift from the 10th up to the drain tunnel, which was 431 ft.,

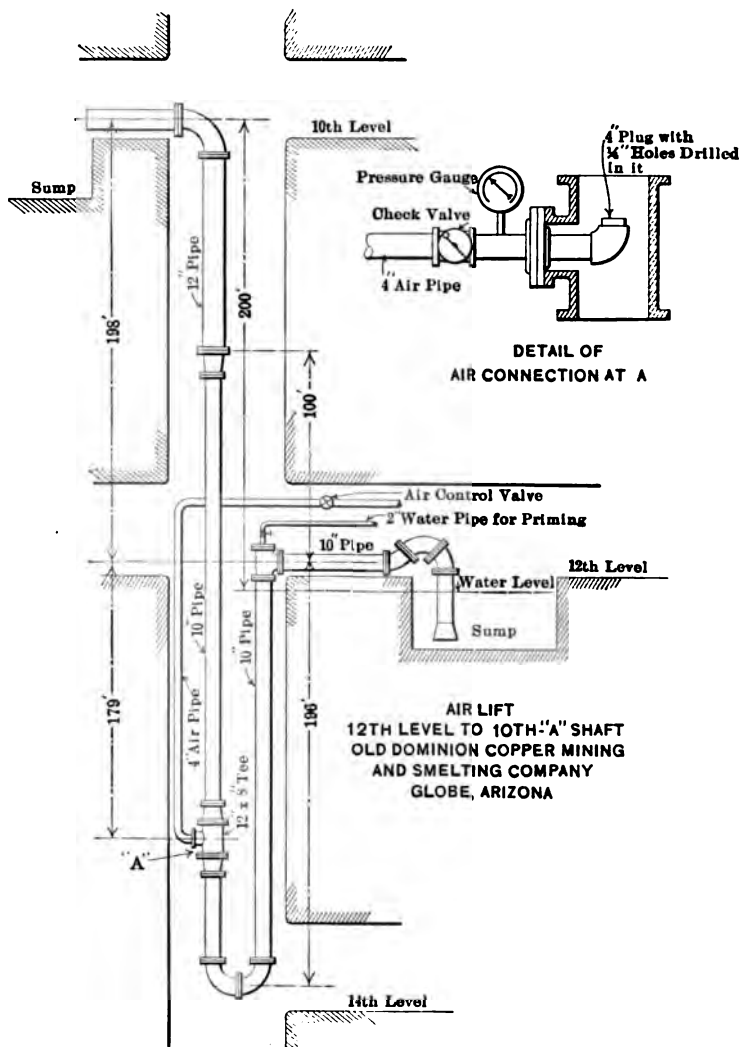


FIG. 9.—ARRANGEMENT OF PIPING ON 200-FT. AIR LIFT.

the submergence was 188 ft., or 30.4 per cent. The top 200 ft. of the column was 12-in. pipe, while the bottom 231 ft., and the two legs were of 10-in. pipe. The air was supplied through a 4-in. pipe at a gage pressure averaging from 90 to 100 lb., but which was always kept as high as possible. This lift delivered to the tunnel 1,233 gal. of water per

minute, with an air consumption of 2,681 cu. ft. per thousand gallons of water delivered. The maximum efficiency showed practically 30 per cent.

During the period of unwatering the 16th level, the two columns on the 18th level Aldrich pumps were converted into air lifts; one delivering to the 14th level and the other one to the 12th. The water at that time stood 36 ft. above the 16th level. These lifts were made up as follows: A 3-in. air line provided with a foot-piece made from 3-in. steel tubing, plugged at the bottom and perforated with $\frac{3}{16}$ -in. holes drilled at a 45° angle, was lowered to the sweeps connecting the columns with the pumps. Both lifts pulled the water through the valves and suction of the Aldrich pumps, and delivered to the 14th level about 1,300 gal. of water per minute, and to the 12th about 1,000 gal. per minute. Fig. 9 gives a sketch of the arrangement of the piping used on the 200-ft. air lift, and Table III shows the result of short tests run on these air lifts.

IV. HIGH-WATER PERIOD

INCREASED FLOW INTO THE MINE

The month of December, 1914, in Globe was the wettest month of the year, over $5\frac{1}{2}$ in. falling in the 31 days, 4 in. of which fell in the 8 days from the 17th to the 25th. The flow of water on the 10th and 12th levels West was about normal to the 20th of the month, and apparently the previous rainfalls had not made themselves felt, but on the 20th day the water originating on the 10th and 12th levels, and which was at that time all handled on the 12th, began to increase from the normal flow of 1,750,000 gal. for 24 hr. until on Jan. 1, we were handling over 3,500,000 gal. per day coming from the 10th and 12th levels West. This, with the more or less constant amount of 2,000,000 gal. from the 16th and 18th levels which was being relayed from the 12th level totalled a flow of 5,500,000 gal. per 24 hr. to be handled from this latter level. By the middle of January 6,500,000 gal. per day was being pumped from the upper and lower levels.

The East water on the 18th level showed no increase whatever during the period the water was rising on the 10th and 12th levels.

Toward the end of January, in order to relieve the 12th level pumps, one of the triple-expansion pumps on the 10th was started, and two weeks later its duplicate on that level was put into commission.

By the middle of February the water was showing in the "C" shaft above the 9th level and the two 10th level pumps were pumping 3,200,000 gal., while the four flywheel triple-expansion pumps on the 12th, aided by the triple-expansion Prescott on that level were pumping 7,000,000 gal. more. All of these seven big pumps were operated by steam gen-

erated at the "A" shaft boiler plant, and the entire battery of six boilers was running overload.

Feb. 26 saw the total water pumped from the mine pass through the 12,000,000 mark. The central power plant provided air for additional emergency pumps on the 10th and 12th levels, and a hurriedly installed 250-hp. boiler at the "A" shaft helped to relieve the overtaxed normal steam generating equipment at that point. By the last day of February, the mine had pumped 270,000,000 gal. in the 28 days of the month, and there was no sign of the water flow slackening. On Mar. 3, the day before the inrush of water, the total amount rose to 12,600,000 gal.

Such, in brief, is the story of the rise of the water to its high point.

INRUSH OF WATER ON 12TH LEVEL, CAUSING FLOODING OF 18TH AND 16TH LEVELS

1. *Account of "Break" on 12th Level*

It became obvious in the first few days of March, that with the water increasing steadily on the 10th and 12th levels, and coming down the "C" shaft in even greater volumes from the 9th level and above, that we had about reached the limit of our available pumping and power capacity, and our only hope was that the flow had reached its height and would gradually decrease. But what actually happened exceeded our most pessimistic anticipations.

On the afternoon of Mar. 4, word was telephoned up to surface that a big flow of water had broken into the new water drift on the 12th level and that the pumps on that level were unable to handle it. A hurried inspection of conditions on the 12th level showed an immense torrent of water coming surging out to the pump station. With all five big pumps running overspeed, the pumping equipment was absolutely unable to cope with the flow. The sumps were overflowing and the water rushing down the pump winze and main shaft to the 18th level. A realization of the volume of water made it clear that the 18th level was doomed to be drowned very quickly. As the 12th level pumps are relays for the 18th level water, it was obvious that if the former were already overtaxed, that it was useless sending any water up from below. Within 30 min. the 18th level quintuplex electric pumps were under water, and by 7 o'clock that evening, the water had risen in the pump winze and main shaft to the 16th level. The motors of the four electric pumps on the 16th level were left in place just as long as possible in the hope that the flow on the 12th would subside sufficiently to save these from loss. When, however, the water started creeping up on the station, hurried attempts were made to get the motors off, but the rise of the water, caused by the overflow from the 12th level was so rapid that within a very short while the work

had to be discontinued. All equipment possible that could be moved, in the form of locomotives, tools, and cars, was transferred to the east side of the mine, 3,000 ft. away, where the water had not yet reached. The torrent of water coming down the main shaft was of such intensity that it became unsafe to operate the cages below the 12th level in the "A" shaft, which is the only operating shaft sunk below the 12th level. In the meantime, a considerable flow of water commenced showing on the 14th level West. This necessitated starting the triple-expansion Prescott pump on that level, which throws direct to the drain tunnel. By 8 p.m. every available pump in the mine above the 16th level was running over-speed, and the water was overflowing from the 12th and 14th levels down to the 16th level, which latter level by late that night was completely submerged to its farthest point east and west.

2. Serious Problems Caused by Inrush of Water

A thorough inspection of conditions in the water country on the 12th level showed that the two most serious matters to be coped with at once were, first, the concentration of all the water possible on the 12th level, and second, the handling of the immense quantity of sand that came surging out with the water. The inspection revealed that the new rush of water that was causing all the damage was coming in from the crosscut marked *CC* on Fig. 4 that connected the old and new water drifts. But how far back toward the old water drift the "break" had come was impossible to tell, owing to the immense quantity of *débris*, varying in size from big boulders down to fine dacite sand, that was being deposited in the crosscut and in the new water drift. As the crosscut became blocked with *débris*, and its outlet, to a partial extent, dammed, the water backed up and found its way along the old water drift to 13-1-11 raise which was used as a ventilating raise for air coming from the "C" shaft. The water was running down this raise to the 13th level and thence through several of the stopes to the 14th level. Reference to Fig. 4 will make the location of these various points clear. It was recognized at once that it was vitally necessary to prevent the water flowing down below the 12th, where, not only were the pumps insufficient to handle any continued large volume of water, but the water could do great damage to the orebodies. It was, therefore, essential that the raise should, if at all possible, be closed or dammed up without delay. The only possible way into the top of the raise would have been through the drift *DD* from the point marked *Y*. Here it was found that the *débris* and sand had come out in such quantities that the drift was completely filled with dacite sand, brought down by the water to within a few feet of point *Y*. This being fully 300 ft. from the raise made it out of all question to reach the objective point from that direction. A similar

filling up of all drifts adjacent to the bottom of the raise on the 13th made it equally impossible to reach the raise within anything like reasonable time on that level, since the removal of sand and rock by mucking out the drifts quickly proved futile, as it simply made room for the further deposition of sand. Being thus effectively cut off from reaching the raise, either at the top or bottom, our main hope was that it would soon choke and block up with sand, of its own accord. All efforts were, therefore, centralized in keeping open the main water drifts on the 12th level from the "A" shaft to the crosscut connecting the old and new water drift so as to insure at all costs a free exit for the water on the 12th, and to prevent any backing up behind the points where the water broke in with the consequent tendency to flow down the ventilation raise to the 13th and 14th levels.

This led up to the second big problem. The water was coming in periodic but tremendously strong surges, carrying immense volumes of sand and rock, the finer sand, despite hastily constructed dams, being carried into the sumps at the pump station. The rushes produced volumes of water and sand that, in addition to the flow already overtaxing the pumping capacity, caused the sumps to overflow continuously. These periodic heavy surges of water were doubtless occasioned by the continual damming up with sand and rock of the main channel through which the water came down from above the 12th level; when the pressure became too strong the water broke through with a rush and then ran steadily for a short time, only to get dammed up again with a repetition of the same performance. With a conception of a very recently formed channel through soft and caving formation, this procedure can easily be imagined. These constant surges and the débris they brought down were the most trying features in the control of the water.

HANDLING OF WATER DURING FLOOD PERIOD AND UNWATERING OF 16TH LEVEL

The night of Mar. 4 found us in the following unenviable position: Water was coming into the west end of the mine from four different sources—on the 10th level, in the vicinity of the "C" shaft, on the level, and from above the 10th level in the shaft itself at the rate of 3,000 gal. per minute, all of which was clear water without sand, and was handled by the 10th Prescott pumps and two sinking pumps.

On the 12th level clear water without sand was coming in from the original source at the concrete dam, while from the most recent source in the crosscut connecting the old and new water drifts the water was pouring out in a torrent into the drift, bringing with it immense quantities of sand and rock. This flow, combined with the clear water, was largely going out to the "A" shaft main pumping station. But there was also a portion of it flowing down the "C" shaft from the 12th to the 14th, while

the remainder of the water which broke in that day was going down the ventilation raise and thence through the stopes to the 14th levels. Water was also coming out of the Zero section on the 12th.

At the pump stations, the four big triple-expansion flywheel pumps and the other emergency pumps on the 12th level were all running over-speed and were pumping at the rate of 7,000 gal. per minute, but making little impression on the flow.

The 14th level Prescott pump was lifting about 700 gal. per minute to the drain tunnel while the sumps on both 12th and 14th levels were overflowing down to the 16th level.

A record of the happenings during the flood period showed for the first few days a gradual tendency to decrease on the 10th level, a disheartening succession of periodic heavy surges, and rushes from the latest water source on the 12th, and an almost constant overflowing of the sumps on the 12th and 14th levels. The water in the Zero country, which we had been trying to drain for some time previous, behaved erratically for several days, coming in big rushes at times, and then subsiding, but after a few days this source gradually drained and in time almost completely dried up.

Each day regular and very thorough rounds of inspection were made to all the points on the 10th, 12th, 13th, and 14th levels, where there were any water flows, and the conditions carefully noted and reported, and any repair work that was necessary to the drifts done.

The longest and most arduous piece of work was mucking out the 12th level drifts leading from the 12th pump station back to where the latest water had broken in. The difficulties were considerable and of sufficient interest to relate in detail.

The first rush of water had brought down, as noted before, a tremendous volume of débris, ranging in size from boulders 2 ft. in diameter, down through small rocks to fine dacite sand. The heavier and coarser material was deposited in the first few hundred feet of drift away from the "break," often as high as 5 ft. in the drift. The coarser sands were carried farther and completely covered the mine car tracks to a depth varying from 1 to 4 ft., while the fine sands really never settled, and were carried in suspension by the water into the sumps.

To keep the main exit in the water crosscut open entailed the shoveling away of the débris as fast as it accumulated at the mouth of the break in order to prevent choking up. At first, so as to have use of the tracks and be able to use mine cars, it was attempted to start cleaning near the pump station, and work in from there, but this early proved futile, for no sooner had 100 ft. or so been cleared, than the shoveling of the sand farther upstream put more sand into suspension, which, in a short time, settled and again covered the tracks and it was soon realized that the objective point never would be reached by this method.

In order to make faster headway, two methods were adopted to get rid temporarily of the sand and rocks and keep the drifts open. A large proportion of the drifts leading from the pump station to the water cross-cut were originally timbered; 2 by 12 planks were nailed horizontally between the posts of each set on either side of the drift as low down as the water would allow, and additional planks nailed on to the face of the posts, forming a space 12 in. wide, 4 ft. long, and about 4 ft. high, into which the sand could be shoveled. By doing this all along the timbered portions of the drifts, considerable sand and débris was removed at the minimum expense. Where the sand had to be carried downstream some distance before shoveling into these temporary storage places, in place of using shovels, a group of men placed about 10 ft. apart took 2 by 12 planks, 5 ft. long, and by holding them about 1 ft. off the bottom of the drift, the sand was automatically sucked under one plank, carried by the water without settling to the next plank, and so on until it reached the desired point. As this method of disposal was only adaptable to the sandy portion of the débris, shallow dams were placed in the drifts opposite suitable crosscuts and platforms placed on the top of these dams. Boats about 10 ft. long and 1 ft. deep were then constructed. These boats were towed upstream several hundred feet on to the platforms where their contents were shoveled into the crosscuts. By dividing the drifts into several sections in each of which two or three boats operated, good headway under the conditions was made. In fact, the absolute impossibility of keeping the tracks passable for cars made the boat system the only one feasible for transporting the sand and rock any considerable distance.

It was an exceedingly unenviable and disheartening piece of work, carried out under very trying conditions. The water was running 3 and 4 ft. high in the drifts where the work had to be done, so that men were working over their knees and often up to their waists in water, the temperature of which averaged 65° F. Conditions such as these are not conducive to very high efficiency, but the men were at all times cheerful and carried on the work in a meritorious manner that cannot be too highly commended. Hot coffee was brewed on surface, and it was one man's job each shift to keep all of the men working in the water well supplied with hot drinks.

The discouraging feature of this drift-cleaning work was that often hardly had a section of track been cleaned when a fresh rush of sand and water would come surging down the drift and again fill everything up.

Near the pump station, the dams were made more frequent, but at the best the fine sand was carried over into the sumps with the water. The sumps were kept cleaned of sand as much as possible by an air ejector which discharged into a mine car.

The water above the 16th level had risen, from Mar. 4 to 13, to a

distance of about 36 ft. in the "A" shaft above the level of the station. Fig. 10 shows the area which this water occupied at each foot of rise above the level, and the rate at which it was unwatered.

As has been described before, air lifts were used altogether for unwatering this level, but they could only be used when the water being produced on the 12th and 14th levels would allow of the pumps on these levels handling any additional flows, but there were no special difficulties encountered in reaching the 16th, other than the necessarily slow progress that was made.

A description of the first few days of the flood is practically a description of the whole month during which mining operations in the mine

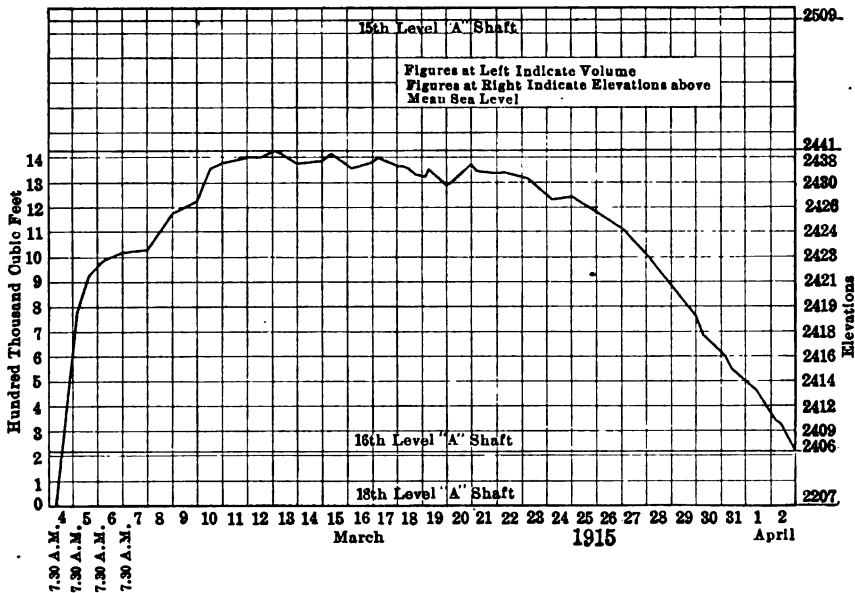


FIG. 10.—RISE OF WATER ABOVE 16TH LEVEL AT TIME OF FLOOD AND PROGRESS MADE IN UNWATERING.

had to be discontinued owing to water troubles. In general, the 10th level water showed a small but steady decrease, and the Zero country on the 12th level became, in time, comparatively dry. The 14th level water from the west country showed a decrease, due more or less to complete choking up with sand of the 13-1-11 raise, and the surges from the new water source on the 12th became less frequent and none were recorded after about Mar. 25. The 16th level was finally unwatered on Apr. 3, or almost exactly a month from the date it was flooded. By the first of April, the flow being handled by the 14th, 12th, and 10th level pumps and air lifts amounted to 12,500,000 gal. per day, and from that time on, the decrease, as is shown in Fig. 6, was comparatively steady but slow. By Apr. 6 the mine was operating again on a small scale, and

steps were being taken to repair the not inconsiderable damage done by the flood.

V. GENERAL CONCLUSIONS

At present (December, 1915), a large flow of water is still running in the channel through which the water broke into the mine, and it is therefore not open for inspection. It is doubtful, even if in time it should turn dry, whether we would be justified in satisfying our curiosity as to the size and general character of the opening at the expense of the risk of tapping additional water, or in making any attempt to block it up to prevent inrushes through the same channel.

With reference particularly to the flow of water from the west side of the mine, judging from our recent experiences and from all the data available over a number of years, indications point to the main mass of water which the mine is draining being held in the diorite formation immediately overlying the dacite. Without doubt this supply of water is fed by the precipitation that falls annually in the district, but from how extensive an area it is impossible to say. During the abnormally wet winter of 1914-1915, it was noticeable that the effect of the yearly portion of the rainfall (up to December) was not felt in the mine. This, doubtless, corresponded to the ordinary season's precipitation, which normally is absorbed by the dry ground and only finds its way into the water basin comparatively slowly. But as soon as the ground was saturated and had absorbed all the moisture possible, all additional rainfall sank and flowed very quickly into the openings in the mine. This, I think, is shown by the flat line of mine water curve up to the end of November, in Fig. 6, when 13 in. of rainfall had fallen since July 1, which period might be called the "saturation" period, and by the appalling rapidity with which heavy rains and floods after that date were felt in the mine. It is possible, in view of the porous nature of the creek bed, that the water flowing continuously in the creek accelerated the seepage into the mine, but with the whole surface area kept in a water-logged condition by the continuous soaking rains, the part played by the creek at this particular time is uncertain. However, as a possible preventive measure, the fluming of the creek over and past the more porous formations, such as the diorite adjacent to the company property and considerably beyond the limit of the old flume, is a logical precaution to take. Although, as noted previously, that portion of the water that the mine is draining on the 12th level is, in all probability, in the diorite, fractures and fissures in the underlying dacite afford easy channels for the downward seepage of the water, and it is in the latter formation that the flows are encountered.

As to conditions in the mine during the period of the rise of water, everything points to the water backing up from the 12th level on the

flat diorite dacite contact, first showing on the 12th, then on the 11th, and then on the 10th levels, and finally in the "C" shaft on the 9th level and above. With insufficient openings for relief, and with practically only one drainage point open on the 10th and one on the 12th level, the pressure became too great and the water broke through at a point on the 12th level connected with a fracture through the dacite where the resistance was weak. It is impossible, of course, to account altogether logically for the water not choosing the more broken up and already wet stopping country in the Zero section, only a few hundred feet away; but probably the zone of the Old Dominion vein, and the system of fracturing in the dacite were the determining factors. With the pressure relieved at the low point on the 12th, the head quickly dropped on the 9th and 10th, but after the first few days following the break, the decrease on the upper levels was very gradual and seemingly not commensurate with the quantity of water pumped from the 12th. This, however, probably only indicates that the filling up of the empty pores of the country was a gradual process carried out on a big scale, and that the complete draining of this same area will be correspondingly slow.

As a preventive measure against uncontrolled flows of water drowning the pump stations, concrete bulkheads in the drifts between the possible sources of water and the pump stations would be of benefit. They should have doors that can be shut tight and fitted with suitable pipes and valves through the bulkheads to allow of no more water coming to the pumps than they can handle. This is essential to avoid the tremendous surges bringing sand and gravel right to the pumps. If properly placed, these would also serve the double purpose of shutting off portions of the mine from the flooded levels, and allowing mining operations to continue in the remainder.

As was natural, the pumps on the 12th level worked at their greatest efficiency prior to Mar. 4, when the flood of water brought sand into the sumps. Although settling ponds were built along the drift, and a final dam near the sump was constructed with a steel screen and burlap covering, it was almost impossible to keep the fine sand away from the suctions of the various pumps, and not only was their efficiency much impaired thereby, but the constant changing of valves caused by sand trouble kept one pump out of commission most of the time, and greatly decreased the total pumping capacity. No valve facing was found that would stand the wear of the sand for long, and the replacing of the big valves on the triple-expansion flywheel pumps, at best a somewhat cumbersome job, was for many weeks a continuous performance.

The valves used in these pumps are 18 in. in diameter and eight in number—four suction and four discharge. The weight of the valve and the valve seat is 300 lb. The valve covers weigh about 600 lb. and are bolted down with 20 $1\frac{3}{4}$ -in. bolts. On account of the great size of the valve,

should particles of coarse sand or other substances get under the valve, holding it up, the sand and water rush through and cut it out very quickly.

Previous to the flood period, we had experimented with numerous kinds of valve facings—brass, steel, fiber, and leather. A record of valve service on the triple-expansion flywheel pumps shows that with clear-water conditions, the brass valves average 37 days in service, the steel valves 70, the fiber 105, and the leather 64. Pump No. 2, which is the clear-water pump and takes its water direct from the concrete dam, has run as long as 180 days without a valve change.

The cost for the various valves are: Brass, \$15.37; steel, \$3.99; fiber \$17.82; and leather, \$3.86, each. The fiber and leather valves have to be discarded when once worn out, while the brass and steel valves can be refaced at a cost of 55 c. and 65 c. respectively. The above figures from normal conditions with these pumps favor the use of the steel-faced valve, but with gritty and sandy water the steel valve gave very short service, and leather, fiber, and rubber were all tried in turn. The fiber and hard leather valves were found preferable, the better grades of fiber lasting from 4 to 6 days, and hard leather about the same length of time.

With the Prescott triple-expansion pumps, the gritty water led to the same valve difficulties, but the work of changing the valves, on account of their smaller size, was infinitely easier. When these latter pumps were running at from 33 to 37 r.p.m. with sandy water, new valves were required about every 48 hr. in both suction and discharge chambers, and we seemed to get better service with the hard rubber valve than with any other kind. At the normal speed and with clear water, rubber valves also give best service in the triple-expansion Prescott pumps, and are often in good condition at the end of two months.

On account of the valve trouble experienced with high efficiency pumps of this type, where there is any likelihood of having sandy water to pump, special attention should be paid to the valve question, and within proper limits more and smaller valves are preferable to fewer and bigger ones; this, on account of their being easier to change, and also the fact that one or two bad valves on a pump where the total number of valves is small, lowers the efficiency quicker than the same number of poor valves on a pump where the total number is much greater; and loss of efficiency is a fatal drawback when so much depends on the amount of water to be handled. On the other hand, with the pumps running at high speed, greater breakage was experienced with the small valves than with the big ones on the flywheel pumps.

If electric pumps are installed for emergency service, they should probably be of the centrifugal type, with lower efficiency, but with the ability to pump sand and grit and needing little attention. The attention fea-

ture is one of the most important in emergency pumping conditions. There seems to be a growing tendency among pump manufacturers to show increased interest in the production of a centrifugal type pump with relatively high efficiency. This is commendable as the market for such a pump is large.

Where there is no surplus steam or electrical power, and where compressed air is available, air lifts will be found more satisfactory, under the right conditions, than sinking pumps run by air power; also infinitely less trouble is caused, air lifts being more or less automatic in their operation. However, a 30 to 40 per cent. submergence is necessary for fairly efficient work. Although as a pumping unit they are uneconomical, they throw large volumes of water, and can be used to very good purpose under certain conditions. They cannot, of course, owing to the submergence necessary, be used down to the lowest unwatering point on a level.

Recent work in unwatering the 18th level, flooded in March, with a 24-in., 5-stage Layne & Bowler deep-well pump showed that it has advantage, where the space will allow, and is much superior to air- or steam-driven sinking pumps for unwatering purposes.

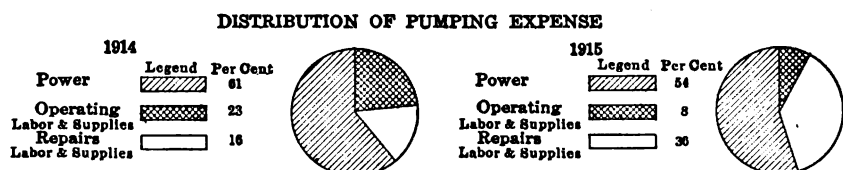


FIG. 11.

Where there is any chance of mud or sand getting into the sumps, care should be taken in the design to allow of their being in duplicate, or at least so partitioned as to admit of thorough cleaning of one section without stirring up the sand in the water delivered to the suctions of the pumps.

Finally, it may be of interest to state some figures on the pumping expense connected with the handling of these large volumes of water. To arrive at an adequate conception of the amount of water pumped from the mine in the first six months of 1915, it is interesting to note that during this period 6,750,000 tons of water was lifted out of the mine from the 10th, 12th, and 14th levels. During this same period, 99,000 tons of ore was hoisted from the mine, or, in other words, for every ton of ore extracted and hoisted, the Old Dominion Copper Co. had to pump out of the mine 68 tons of water, a proportion of water to ore that is, obviously, not conducive to cheap mining. During the first half of the year, the pumping expense amounted to 29 per cent. of the total underground, hoisting, and surface expense. Details of the pumping costs during the flood period were so abnormal as to have no particular interest, but it may be stated that of the total pumping expense, the operating labor and

supplies amounted to 10 per cent., power to 54 per cent., and repair labor and supplies to 36 per cent. In the previous year, 1914, operating labor and supplies amounted to 23 per cent. of the total, power 61 per cent., and repair labor and supplies 16 per cent. This is shown graphically in Fig. 11. The repair expense consists of repairs, labor and material for steam lines, water columns, pumps, sumps, and drain tunnel, as well as the installation of emergency pumps, air lifts, etc. In the power expense, which is the heaviest individual item of the pumping costs, fuel oil is practically 87 per cent. of the total, which, during the period from Jan. 1 to June 30, 1915, would make the fuel oil expense 47 per cent. of the total pumping costs.

In conclusion, my thanks for the compilation of certain portions of the data used in this paper are due to I. H. Barkdoll, Mine Superintendent, H. L. Norton, Chief Engineer, Charles Mendelsohn, Mechanical Engineer, and G. N. Bjorge, Geologist.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

The Application and Earning Power of Chemistry in the Coal Mining Industry

BY EDWIN M. CHANCE,* WILKES-BARRE, PA.

(Arizona Meeting, September, 1916)

DURING the last decade many conditions have been encountered that have materially increased the cost of the production of coal. As in most cases it has not been practicable to increase the selling price of such coal sufficiently to insure the necessary margin of profit, the mine management has been forced to avail itself of what might be termed unusual means that costs might be held at as low a figure as possible. The so-called efficiency engineer has been called upon, costly power plants have been built and machinery has been installed to replace manual labor wherever possible.

Another aid used recently to secure economies through the more intelligent production of coal, is the chemist. As the coal industry, barring the production of byproduct coke, does not at once give evidence of its need of the services of the chemist, the writer will endeavor to point out a few of the services that this branch of the staff has been able to render.

The preparation of coal in the past has been carried on entirely upon an empirical basis, that is, the criterion adopted has been an ocular inspection. Unfortunately, the appearance of a coal and especially of an anthracite, has but little relation to its fuel value, and as long as coal is purchased for the heat that can be obtained from it by its combustion, the quantity of such heat that is purchased must be one of the principal desiderata in selecting or preparing a coal. Were coals purchased for use as bric-a-brac or for exhibition in museums it might be both wise and just to purchase them upon their appearance. After 4 or 5 years of investigation the fact has been established that coals of very inferior appearance, and I now have special reference to anthracite, might and often did possess heating and burning properties superior to those possessed by bright coals which in the past had commanded considerable and uniform premiums in the market. This being the case the question arose as to whether the trade could be convinced of this fact. While at first some difficulty was experienced in establishing the quality of such dull coals, still it was found that the real obstacle to the sale of these

* Consulting Chemist and Engineer.

materials in the past had been the fact that the coal producers' own sales departments felt convinced of the inferiority of this type of fuel and when complaints were registered they were given ready credence. Upon convincing the sales department of the quality of this type of fuel, but little difficulty was experienced in securing a ready market for it and it was even found possible to secure premiums for certain of these dull coals. It is readily understandable that the only available means whereby a true appreciation of the quality of a fuel can be arrived at is by its chemical analysis or by a service test. The cost and unreliability of service tests have led to their general abandonment, hence the chemical examination of a fuel has been found to be perhaps the most ready and practicable method for determining its real position in the scale of usable fuels. Now when we consider that in the past hundreds of thousands of tons of dull and unpromising looking coal have been gobbled at a positive cost to the coal producer, with the ever present danger of gob fires, that many other hundreds of thousands of tons of this material have been sent to the rock bank as refuse, and that because of the similarity of the specific gravities of the dull and bright coals, the difficulties and inefficiencies of coal preparation have been enormously increased by the attempted removal of this material, it will be seen how enormously profitable the utilization of this excellent fuel has been. In order to give an adequate idea of the relative worth of the dull and bright material from the same veins, I will quote the following average analyses:

Size	Test Number	Color of Ash	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulphur, Per Cent.	Heating Value per Pound Coal as Received, B.t.u.	Heating Value per Pound Dry Coal in B.t.u.
Dull coal.....	C-502	Red	2.98	6.39	82.24	8.39	0.67	13,110	13,510
Bright coal...	C-503	Red	3.70	5.67	81.16	9.47	0.71	12,890	13,390

These analyses are in both cases the average of a number of analyses of characteristic material of both types, and are in no sense selected. In the past this dull material, test No. C-502, has been condemned as bone. Thus by a systematic chemical investigation of the various steps in the preparation of coal undiscovered inefficiencies are revealed and their correction is made possible.

In the efficient operation of boiler plants it is necessary to secure as soft and non-corrosive a boiler water as possible. While the most desirable method is to secure a supply of water that is naturally soft, still in the mining industry it is frequently necessary to use water that is both incrusting and corrosive. To meet these conditions a host of so-called boiler compounds has been foisted upon the unsuspecting mine manager. While certain of these compounds are worthless or of little value, others

have considerable merit in special cases. Unfortunately, however, their use is indiscriminately recommended by the vendor, and a boiler compound that might prove efficient with waters of a certain type will prove of little value under different conditions. Moreover, the cost of these compounds is out of all proportion to their merit and a grave and positive risk is run of doing the boilers infinite damage by the indiscriminate use of such nostrums. I recall at this moment the case of one new boiler house equipment of several thousand horsepower capacity that was so damaged by 3 months' use of a highly recommended water softener, even though the water in this case was not very hard, that it required over a year and a half of patient and costly effort to put the plant back in a condition having the semblance of efficiency. Now a competent chemist can readily recommend a treatment for such boiler feed waters that will meet the requirements of the particular case in question both cheaply and efficiently.

By reason of inadequate storage facilities at many mines, a prolonged drought will occasionally require the use of mine water as boiler feed. The writer recalls one occasion when a large number of collieries of one company were operated with such feed water, containing in many cases as much as 10,000 parts per million of free sulphuric acid and 35,000 to 40,000 parts per million total acidity. These plants were operated successfully over a period of months with such feed water, though of course many difficulties arose and much care was required, while the surrounding coal-mining companies, not possessing expert chemical assistance, were in many cases required to cease operation.

The subject of lubrication is one that has in the past been favored by the so-called efficiency expert. It is, therefore, with a sense of some diffidence that the writer approaches this subject. As conditions governing colliery lubrication are so very different from those governing the lubrication of the mill or industrial plant, it has been found that the lubrication secured by the use of lubricants recommended by the vendor has in the past been unsatisfactory and somewhat costly. Some years ago the writer conducted numerous field and laboratory tests in order to establish just which lubricants would best meet coal mine conditions. After these points had been well established an effort was made to go into the open market and, by specification, purchase the material desired. While this plan was successful, still it was found that almost continual testing of the material supplied was required, as the bidders purchased their materials in the open market and hence there was little or no uniformity of supply. Those bidders not purchasing oils in this way were, in many cases, refiners handling such a diversity of crudes that practically the same complaints arose. It was found that the independent refiners of Pennsylvania petroleum produced the lubricants required, and since the materials were produced by them and their range of production was

rather limited, little difficulty was encountered in securing uniform products from them. Moreover, by concentrating the purchases for a number of coal producers upon one refinery, it has been possible to secure most advantageous prices while at the same time the quality of materials supplied is directed by the purchaser and not by the vendor. In other words, the oil to meet any special condition is purchased from the refiner without consulting him as to its use. Hence there is little or no probability of his attempting to substitute a material that he may consider almost as suitable, as he has no knowledge of the use to which the material in question is to be put. It is an axiom that given a maximum price that a purchaser will pay, the average vendor will endeavor to supply the material that costs him the least but that will meet the condition in question. In purchasing lubricants as indicated above this condition does not arise, as the oil for the purpose in question is selected without considering its price and further, since the vendor is not consulted in the matter, the best oil that can possibly be obtained for the particular purpose is secured. It may seem that the cost of such a procedure would be prohibitive, and this would be the case were these materials purchased at the scale of prices that usually obtains. By means of the coöperation mentioned above, however, prices have been secured that are so advantageous that the highest grade of product that it is possible to secure, irrespective of price, is generally purchased at a lower price than the product of mediocre quality that was formerly used. In this way the actual first cost of lubrication has been reduced from 30 to 50 per cent., while the actual efficiency, though less readily determinable, of the lubrication secured has been greatly increased.

The chemist has been found to be almost indispensable where a mine fire is being extinguished by cutting off its air supply, for the chemical analysis of the air from the fire zone makes it possible to form an intelligent opinion of the progress of the extinction and the air-tightness of the dams or fire walls built to control the fire.

Many more instances than those briefly touched upon might be quoted and those that have been quoted might be developed in greater detail. Such a dissertation, however, would be burdensome and would fall without the purpose of these notes.

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A Combined Hydraulic and Mechanical Classifier

BY M. G. F. SOHNLEIN, MACHACAMARCA, BOLIVIA, S. A.

(Arizona Meeting, September, 1916)

In a Bolivian tin concentrator an appliance was needed to furnish a suitable product for fine jigging from a pulp of the following composition:

Mesh	Per Cent.
+ 20	8.0
+ 40	36.5
+ 60	9.0
+ 80	10.5
+100	5.5
+150	4.5
-150	25.0
	99.0
Loss	1.0

In the jigging process, particles of cassiterite as small as 0.1 mm. can be recovered, provided they are not present in excess, that is if the material treated on the jigs contains sufficient coarser sands to keep the interstices between the grains open. Removing the fine material from the pulp by screening, to prepare the jig feed, is impractical, because that means the use of an 80-mesh screen or finer, and is not as effective as hydraulic classification. A screen would eliminate mineral grains from the feed which can be recovered by jigging, and in addition would throw more gangue and low-grade middling on to the jig.

Formerly a one-spigot Richards vortex classifier had been used to accomplish this separation, but the tangential openings through which the hydraulic water enters became clogged by fine vegetal matter with which the water was contaminated, and which it was impossible to remove from the water. Consequently the work of the classifier was imperfect and a good deal of slime was sent to the jig. When designing the plant referred to in this paper it was decided to use a hydraulic classifier as shown in Fig. 1. The essential features had been copied from the Anaconda classifier.¹ This remodeled classifier consists of a truncated pyramidal wooden hooper A with an attachment of cast iron

¹ *Trans.*, vol. xlvi, p. 277 (1913).

bolted to it at the bottom. The heavier and larger particles settle and pass through the $1\frac{1}{4}$ -in. nipple *C* which is screwed into a flange and laid loose in a recess turned in the casting *B*. The hydraulic water enters through the $1\frac{1}{2}$ -in. pipe *D* which is screwed into the casting. The distance of the pipe from the exit of the nipple *C* can be regulated by screwing the pipe up or down, a short hose connection being provided between the valve *E* and the mill water piping to make this flexible.

The work of this classifier was satisfactory if sufficient hydraulic water was used, but under these conditions a good deal of material that should have gone to the jig was carried into the overflow. The difficulty could have been remedied by again submitting the overflow to hydraulic classification in a second pocket, but this was objectionable since it

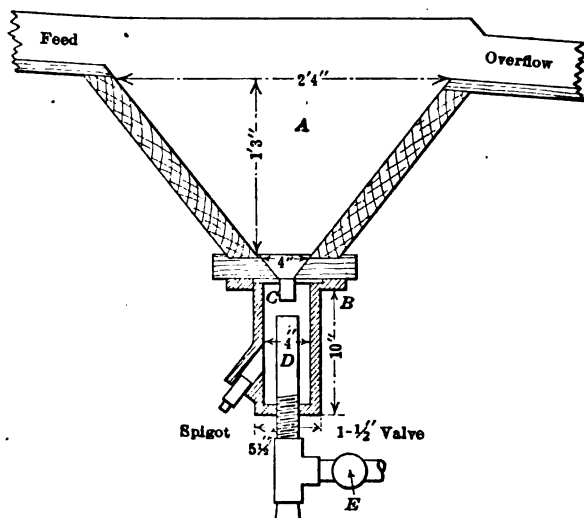


FIG. 1.—SECTIONAL ELEVATION OF HYDRAULIC CLASSIFIER.

would have added another quantity of hydraulic water to the pulp, and diluted the overflow too much. The classifier was therefore run with less hydraulic water so that its sand product contained some slime. The separation effected by the jig was not quite satisfactory under these conditions since the large amount of fines in the feed made the beds pack in hard banks, producing a low-grade concentrate and causing free mineral to go into the tailing.

The desideratum in this instance was to obtain separation of the part of the sands in the pulp under hindered-settling conditions, or nearly so, without diluting the fine material with too much water. The sand product from a mechanical classifier, although practically free from slime, is the result of free settling of the pulp, and therefore is not as desirable as the sand separated by hindered settling in a hydraulic device. I

therefore designed an apparatus which combines the advantages of the hydraulic and the mechanical classifier. Having previously used a paddle-wheel of the Fleming type for dewatering, this machine was selected as a mechanical classifier in preference to the more generally used types on account of its compactness and its low first cost.

As shown in Fig. 2, the device consists of a narrow wooden trough in which the paddle-wheel revolves. Details of the construction of the

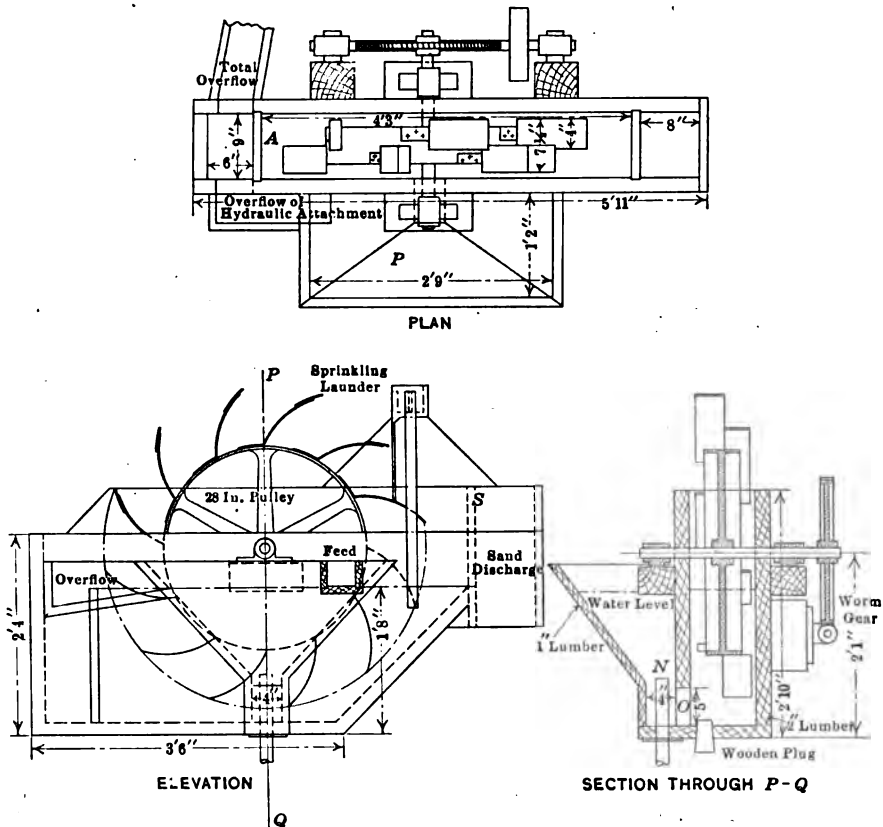


FIG. 2.—COMBINED HYDRAULIC AND MECHANICAL CLASSIFIER.

classifier can be easily gathered from the accompanying drawing. The tips of the blades are protected by $\frac{1}{4}$ -in. iron strips fixed with two counter-sunk rivets. The strips last about four months and are replaced at small cost. The blades are cut from $\frac{1}{4}$ -in. plate and bent in the fire. They are fixed to the rim of the pulley by three 1 by $\frac{3}{8}$ -in. machine bolts. The holes in the rim and those in the blades are bored from a template and spare blades are kept ready so that little time is lost in making renewals. The wheel is driven at the rate of 3 r.p.m. by a simple worm

gear—in this case taken from an old vanner—which is directly belted to a line shaft. The feed is introduced into the trough near the bottom at *O*, after having been submitted to hydraulic classification in the pyramidal attachment *P* in which an ascending current of water flows out of the pipe *N*. Consequently the material which is undesirable in the jig feed is floated off with the use of less hydraulic water than needed in an ordinary hindered-settling classifier because the slime that comes with the sand into the trough is separated by the paddle-wheel and flows over at *A*. It is evident that classification in the hydraulic pocket does not entirely take place under hindered-settling conditions, because in that case the sand treated by the wheel could not contain any slime. The amount of hydraulic water used is such that the dilution of the overflow does not exceed the required density and therefore the classification cannot be perfect. However, the most objectionable factor, namely the presence of slime in the jig feed, is to a large degree eliminated by the subsequent action of the paddle-wheel. The moisture in the sand discharged by the wheel can be regulated in two different ways, (1) by raising the level of discharge, and (2) by increasing the distance between the tips of the blades and the discharge.

The discharge level of the sand product can be varied by inserting strips of wood of different heights into the slots marked *S*. To obtain practical results the sand is discharged at a point $1\frac{1}{2}$ in. above the overflow with 50 to 55 per cent. moisture, but with the level of sand discharge raised to $2\frac{1}{2}$ in. above the overflow, and the discharge edge spaced from the wheel as shown in the drawing, the product contains but 30 to 35 per cent. moisture, but under these conditions the capacity decreases out of all proportion. If the machine is overfed, the sand piles up so high at the discharge that part of it remains lying on the paddles, and falls back into the trough at the other side, finally choking the classifier.

The second method gives a drier and cleaner product, because the sand is piled up in front of the wheel until it reaches sufficient height to slide over the discharge edge, and nearly all the water drains back into the trough. There is no objection to placing the overflow rim in the trough close to the paddles, which makes the machine more compact. I formerly tried to obtain a more thorough settling by leaving a distance of about 10 in. between the tips of the paddles and the overflow, with the only result that the space between became tightly packed with fine sand, and the actual overflow was formed at only $\frac{1}{2}$ in. from the tips.

A spray is provided to remove the adhering grains of sand from the blades of the wheel, and after having cleaned the blades, the water falls on the pile of sand lying in front of the wheel and gives it a final wash. The slower the wheel revolves the cleaner will be the sand product, because then every blade brings up slightly more sand than it can deliver, and part of the material falls back into the trough to be shoved up again by the next

blade. Therefore the sand is repeatedly turned over before being discharged, which materially assists in removing the slime.

The classifier can easily handle 60 tons per 24 hr. of dry pulp with a specific gravity of 3.5 to 3.8. Screen tests made on the products are shown below:

Mesh	Sand Product Per Cent.	Overflow, Per Cent.
+ 20	11.0	0.3
+ 40	42.0	5.4
+ 60	13.5	12.5
+ 80	16.3	11.0
+100	8.0	9.3
+150	2.5	11.5
-150 sand	4.2	48.2
slime	1.4	
	98.9	98.2
Loss	1.1	1.8

These figures show that there is no separation at a definite screen size, but this can not be expected when classifying a pulp that contains particles of cassiterite with a specific gravity of over 6.5 and particles of gangue with a specific gravity less than 3.0. The plus 60-mesh material in the overflow is all very low-grade, and the minus 80-mesh in the sand product is the richest material in this product.

The jigging practice has considerably improved with the better adapted feed and the grade of the concentrate has been raised from 66 per cent. metallic tin to an average of 70 per cent. and more. The concentrate from the jig, which is of the Harz type making a concentrate on three sieves, has the following composition:

Mesh	Per Cent.	Metallic Tin, Per Cent.
+ 20	20.9	71.7
+ 40	55.3	72.2
+ 60	10.4	71.0
+ 80	4.6	65.6
+100	2.7	66.2
+150	3.6	70.1
-150	1.3	68.3
Total and average	98.8	71.36
Loss	1.2	
	100.0	

The feed to the classifier contained in this instance 3.5 per cent. tin, the sand product or feed to the jig was not assayed, but carried probably somewhat more cassiterite than the original pulp.

When working on a richer feed of about 13 per cent. tin, the concen-

trate from the jig contains a much larger percentage of fines as is shown below:

Mesh	Per Cent.	Metallic Tin, Per Cent.
+20	10.0	71.8
+40	36.5	72.1
+60	16.5	72.7
+80	12.0	72.0
+100	6.5	69.8
+150	7.0	69.2
-150	10.0	70.3
Total and average	98.5	71.98

I will now explain why in this case jiggling is used for the treatment of a pulp that could be handled by tables. When dressing an ore assaying less than 5 per cent. tin on tables it is impossible, even with the best classification, to obtain at one operation a concentrate assaying better than about 63 per cent. tin, whereas a jig gives a product of a considerably higher grade, and once it has been properly adjusted does not need much attention, even with a varying amount of feed. When working on richer material, a table does not produce a concentrate of more than 66 per cent. In tin dressing the grade of concentrate produced is of paramount importance, on account of heavy freight charges to the smelters and the reduced smelting charges on high-grade products. It is therefore advisable in every case to recover as much concentrate as possible on jigs.

The whole construction of the classifier could, of course, be improved, especially as regards the form and overflow of the hydraulic attachment, but since the machine does the work expected of it, it is not worth while to make changes. The power required to drive the wheel is under $\frac{1}{6}$ hp. The only attention required is to lubricate the bearings occasionally and to adjust the water valves, if the quality of the feed varies. The machine has been working since the beginning of July, 1915, and at the date of writing (November, 1915) the tips of the paddles have just had to be renewed. One paddle has broken near the rim of the pulley, but this was probably due to poor material or perhaps because the paddle after bending had been cooled too quickly.

After writing this I noticed the classifier proposed by A. E. Drucker in the *Mining and Scientific Press* of Oct. 16, 1915, which is essentially the same as the paddle-wheel with hydraulic attachment. There is no reason why the latter machine could not be used with advantage to prepare the feed for a regrinding mill.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Gold and Silver Deposits in North and South America*

BY WALDEMAR LINDGREN, BOSTON, MASS.

(Arizona Meeting, September, 1916)

I. INTRODUCTION

At the time of the discovery of America the old world had a scant supply of the precious metals. Both the northern and the southern part of the new continent proved wonderfully rich in gold and silver and its treasures were eagerly looted; though the looting has lasted four centuries, the mines of its mountain chains are far from being exhausted. Even the later discoveries in Australasia and eastern Siberia could not rob the Western Hemisphere of its position as the greatest gold and silver producing region of the world, though finally the developments in a narrow and circumscribed area in South Africa wrested from the Americas their supremacy in the production of gold.

Nevertheless the history of the two parts of the great western continent has been strikingly different. At first the Spaniards extracted vast treasures of silver from Mexico, Peru, and Bolivia, while Colombia and some placer deposits in Peru yielded a smaller quantity of gold. A couple of centuries later, a stream of gold began to flow from Brazil, the silver production from the countries mentioned above continuing strong in the meanwhile. Later on, the yield of South America diminished, but to offset this there began a wonderful series of discoveries in North America. The gold fields of California astonished the world; and when the cream of these had been skimmed off there began a no less amazing development of the Central Cordilleran gold and silver districts, which soon made the United States the greatest producer of the precious metals. Aided by ever improving technique, extensive exploration, and a system of railroads, the yield was maintained and increased. Still later followed the discoveries of the gold fields of the arctic region and silenced those who had maintained that the zenith in gold production had passed. Recently the province of Ontario in eastern Canada

* Read at the 2nd Pan-American Scientific Congress, Washington, D. C., Jan. 3, 1916.

rose unexpectedly with offerings of the richest silver ores the world has known, and with new and at first doubtfully accepted gold fields.

Chile and Bolivia in the middle of the last century added some rich silver mines to their long list of mining districts, and later placer gold began to be extracted in large quantities from the Guianas, but on the whole no such sensational finds were made in the southern continent as had marked the recent history of the northern part, and in many regions the mining of the precious metals fell into a rut, the

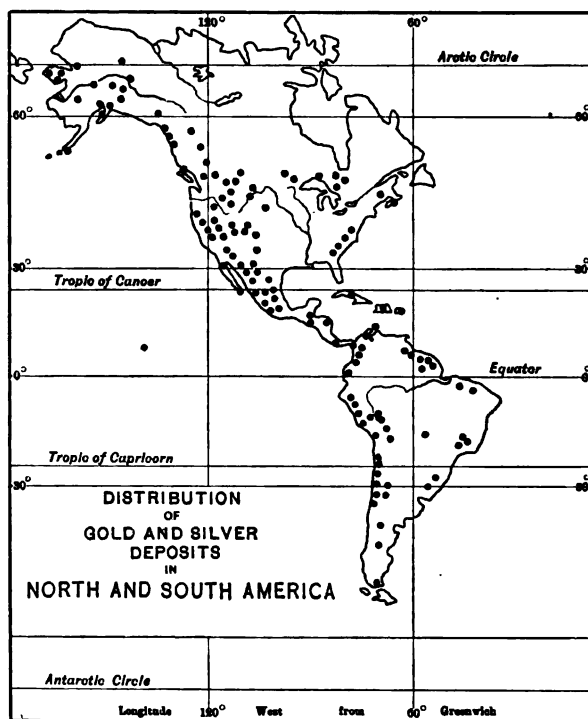


FIG. 1.

production being barely maintained or diminished slowly. The latest events indicate an awakening, and a stimulus under the influence of which the production of South America is gradually increasing. Large amounts of silver are extracted in the copper from operations on a large scale, and dredges dig up the gold of Colombia and Tierra del Fuego.

It cannot be doubted that the total yield of the northern continent of gold and silver is larger than that of the southern part. A glance at the table in the Appendix will show that this difference is strongly emphasized at the present time. During the last decade the gold production of North America had a value of \$1,338,268,000,

while South America yielded only \$125,000,000. For silver, South America statistics are in less satisfactory shape, but the compilation shows that while in 1913 North America produced this metal to the value of \$99,476,400, South America's mines yielded less than one-tenth of this huge sum. Fig. 1 shows the approximate distribution of the gold and silver deposits of the two continents.

There are no better prospectors in the world than those of some South American countries, and we may rest assured that a great percentage of possible discoveries has already been made. Yet no one who has studied South American mining districts can fail to see the possibility of a more extended production than at present, even while realizing the difficulties of climate, altitude, transportation and lack of adequate available capital.

The purpose of this paper is to call attention to the geological features that govern the distribution and richness of the precious metal deposits of South America, to compare them with those of North America, and to classify them according to geological affiliations.

II. GEOLOGICAL FEATURES

A slight acquaintance with the geographic features of the two parts of the American continent suffices for the realization of their essential similarity. The two landmasses, elongated from north to south, have a wide eastern part occupied by fertile plains, hilly country or low mountain ranges, and a narrower western part, with the rough topography of an almost continuous high mountain chain closely following the Pacific Coast, narrow in South America, broadening in North America. This is one of the great earth features, and is known as the American Cordillera. In South America it is also known as the Andes. Considered on a large scale its build is simple, though in detail it is diversified by two or more parallel ranges, by intermontane high plateaus or valleys, and by volcanoes, many of which are active.

To the geologist this difference of east and west is sharply accentuated, for he knows that the Atlantic side represents the area of quiet where strong mountain-building forces have rested for millions of years—since the close of the Paleozoic era—while the leveling agencies of erosion and sedimentation have been at work. He knows that the western margin marks the long strip of weakened earth crust along which tangential stresses have played since early Mesozoic times. These stresses culminated in the early Tertiary times causing folding and violent thrust faulting, as if an irresistible force had forced a wrinkle in the earth's crust eastward. These Cordilleran disturbances reach their maximum along the inner eastern edge of the chain. To some degree they still continue, accompanied by uplifts and depressions. Lava flows have been poured out in great volume from volcanoes along the Cordilleras, especially on

the western side, and this has continued at least from the early Mesozoic to the present time. At the same time masses of molten rock have been forced up from great depths into the rocks nearer the surface, and cooled there to granites and diorite porphyries without ever reaching the surface, though through gradual wearing away of the covering rocks many such masses are now exposed at the surface.

Almost all primary gold and silver deposits have been formed during or shortly after epochs of volcanic or intrusive activity. Secondary deposits are derived by the disintegration and concentration by water of such primary deposits. They are called placers or alluvial deposits and are usually cheaply and easily worked.

On the American continents the primary gold and silver deposits date from two widely separated ages. The first period is geologically very ancient and belongs to the pre-Cambrian or early Paleozoic; its deposits are thinly scattered over the entire continental area, but are at many places covered by later rocks. The second period is much more recent, and belongs to the late Mesozoic and the Tertiary. Its numberless deposits were formed during the great igneous activity which accompanied the building of the Cordilleras and are thus confined to the western or Cordilleran part of the continents in which area the deposits of the older period are rare because capped by later sediments or flows.

Placers may be formed from deposits of either period.

1. DEPOSITS OF THE EARLY PERIOD

Naturally, the deposits of the early period are best observed in the great eastern expanse of the continents where the early rocks are often splendidly exposed. Gold is the principal metal and is always accompanied by quartz gangue. The deposits bear evidence of having been formed at considerable depth and high temperatures. While the majority of these occurrences are poor, yet great richness may be found in small areas, and the purity and coarseness of the gold is favorable to the formation of placers, especially in temperate or warm climates. Wherever continental ice sheets have covered a region, as in Canada, they have almost invariably ground up and scattered the placers.

North America

In North America deposits of this kind are formed in the southern Appalachian States, in South Dakota, in Quebec, in Nova Scotia, and in Ontario. In the latter province the recently discovered Porcupine district presents a case of extraordinary richness, the annual production being now over \$4,000,000. The celebrated Homestake mine in South Dakota is working on a pre-Cambrian replacement deposit in form of thick lenses of altered schist with free gold. While containing only

about \$4 per ton, the ores yield annually over \$5,000,000. A number of scattered deposits of this kind are found in the Cordilleran region of the United States, but they contribute only small amounts to the total production. The most important occurrence is the copper deposit of the United Verde mine in Arizona. Its copper bullion yields a considerable amount of gold and silver.

Compared to the deposits of the Cordilleran or younger period in North America the yield—both total and annual—is small. Out of an annual gold production of about \$130,000,000, the sum to be credited to the old group of deposits is at present (1913) not more than \$10,000,000.

Very little silver is obtained from the gold deposits, but a small amount comes from the copper deposits of the Lake Superior districts. Until the discovery of the Cobalt district in Ontario the proportion of silver in the eastern region to the total output was even smaller than that of gold; but the native silver yielded by this district (almost unique in America) has changed this so that the old deposits of the East are now credited with about 800,000 kg. out of an annual production for North America of over 5,000,000 kg. The great output of the Cobalt district emphasizes again how highly the precious metals may be concentrated within small areas.

South America

In South America we find extremely similar geological conditions, but here the older group of deposits yields decidedly more gold than that furnished by the belt of the Andes. On the other hand, the silver production of the older deposits is insignificant. A somewhat more detailed review will perhaps be acceptable. Fig. 2 shows the distribution of the deposits in South America.

Gold deposits of the older type are known from Venezuela, the three Guianas, Brazil, Uruguay and Argentina. Except in the Guianas they do not form continuous belts, but rather a series of scattered occurrences separated by barren ground or by younger transgressing fluvial or marine deposits. South of the latitude of Buenos Aires the deposits, if existing, are covered by the Tertiary Pampas formation or by lavas of the same age.

The northeastern region extends 650 miles from the Yuruari basin in eastern Venezuela to the Franco-Brazilian border of the Guianas. The occurrences worked are mostly placers, to the formation of which the conditions are very favorable; but quartz veins or mineralized dikes have also been exploited. The best example of the veins is furnished by the great Callao mine in Venezuela, which, during its life of 30 years (1865–1895), is said to have yielded \$28,000,000 in coarse gold. Active exploitation of the placers and some veins is going on in the three Guianas at present, the French colony yielding the greatest amount. In 1912 the

production of this belt was about \$5,000,000 and for the last decade it has not been less than \$4,000,000 in any one year.

The primary veins from which the placers have been derived are contained in pre-Cambrian schists, diorites, diabases, granites and granite porphyries.

The gold belt seems to continue to the southeast beyond the boundaries indicated, for it is reported that gold occurs in the provinces of

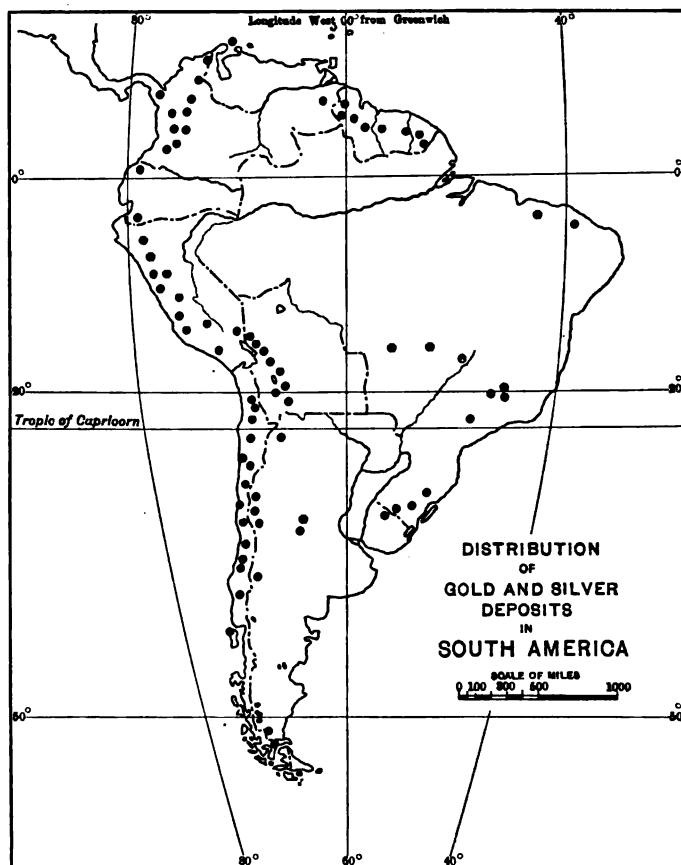


FIG. 2.

Para, Maranhao, and Ceara, in Brazil, beyond the delta of the Amazon River. To the south follows a broad, barren interval until we come to the gold deposits of southern Brazil, in the states of Bahia, Minas Geraes, Sao Paulo, Parana, and Rio Grande do Sul. Of these the state of Minas Geraes is by far the most important. Even in the far western part of Brazil, at Cuyaba in Matto Grosso, occur placers said to be derived from older deposits similar to those of Minas Geraes.

It is almost forgotten at the present time that the placers of southern Brazil yielded heavily in the 18th century, particularly from 1700 to 1775, and this production was particularly welcome at a time when the gold from the Americas seemed exhausted and the treasures of the northern Cordilleras were as yet undreamed of. During the period named, these placers yielded from \$1,000,000 to \$2,000,000 annually. The total yield during the 18th century is variously given from \$200,000,000 to much higher figures. After this period the production languished, but a few quartz mines continued to be operated and a little placer gold was washed. At the present time Brazil maintains its output of gold at from \$2,000,000 to \$3,800,000 but this is practically derived from three deep mines in Minas Geraes, of which the Morro Velho is the most important, besides having the distinction of being the deepest mine in the world (vertical depth 5,800 ft.).

The deposits are quartz veins of a deep-seated type, allied in places to pegmatite dikes. They occur in part in Archean schists, gneisses and granites, but most of them are found in a thick sedimentary series of schists and quartzite, which is older than the Cambrian but overlies the Archean. This series contains no intrusives, except some pegmatite dikes, and the Brazilian veins are in this respect markedly different from most other pre-Cambrian occurrences. It is believed that igneous intrusions took place in the rocks underlying the pre-Cambrian sediments and that only pegmatitic dikes and quartz veins reached up into the covering series.¹

Similar geological conditions prevail in Rio Grande do Sul, beyond which the gold-bearing region continues into Uruguay, where the most southerly mines are found near Cufiapirú. Uruguay yields annually up to \$100,000 in gold.

The most southerly representatives of this older class of gold deposits appear in the Sierras of the Pampas, for instance, in that extending from San Luis to Cordova in Argentina. The old crystalline schists, granites, and pegmatites here emerge from under the Pampas formation and the Permo-Triassic beds, and contain deposits of tungsten, gold and silver, but the latter two metals do not count in quantities sufficient for economic mining.

While it is possible that some deposits of this kind occur in the pre-Cambrian of the Andean region, which is exposed in Colombia and in the northernmost provinces of Argentina, it is improbable that they contribute perceptibly to the total production.

To sum up: The old gold deposits yield the total production of Venezuela, the Guianas, Brazil, and Uruguay and at the present time contribute to the gold production of South America about \$8,500,000,

¹ E. C. Harder and C. K. Leith: *The Geology of Central Minas Geraes, Brazil*, *Journal of Geology*, vol. xxiii, pp. 341 to 378; 385 to 424 (1915).

or not far from the amount extracted from the same class of deposits in North America.

2. DEPOSITS OF THE LATER PERIODS

General Features

From Cape Horn to Alaska the gold and silver deposits are formed under similar geological conditions, and are of the same general geological age. It has already been emphasized that they are products of the igneous activity which has accompanied the rise of this gigantic mountain chain.

They were formed within several epochs, but all of them lie between the earliest Cretaceous and the present; that is, they are late Mesozoic, Cenozoic or Quaternary in age. They were formed, on the whole, nearer to the surface than the old deposits of the pre-Cambrian, or at least under conditions of more moderate temperature. Many of them, indeed, were formed very close to the present surface. Following intrusions or lava flows, hot waters loaded with gases and metals of igneous origin rose toward the surface, and, in cooler regions of the crust, deposited their load of metals. In part the gold and silver occur in minute quantities associated with copper and lead minerals, and are recovered from the base bullion. Much silver is obtained in this manner, but most of the gold is derived from gold quartz deposits, properly speaking, or from placers caused by the wearing down by erosion of these deposits.

North America

It is difficult indeed to give in a few paragraphs even an approximate idea of the gold and silver deposits of the North American Cordillera. The annual yield of the region is enormous, attaining now \$130,000,000 in gold and nearly \$100,000,000 in silver.

A great gold producing belt lies along the Pacific and reaches from California to Alaska, with local interruptions. These are the oldest deposits of early Cretaceous age and they have yielded vast placer or secondary deposits. The annual production including the placers, is not less than \$40,000,000. Geologically they are connected with the intrusion of dioritic rocks, an intrusion extending like a gigantic dike along the Pacific Coast mountains.

Throughout the interior part of the Cordilleran region are numberless smaller intrusions of granitic or dioritic rocks, or of the porphyries of these rocks, most of them of earliest Tertiary age, some a little earlier, others a little later. Aureoles of gold and silver veins surround these intrusions, and contribute from numerous centers in the interior Cordilleran region to the total production.

Contact-metamorphic deposits, formed where limestone beds have adjoined the igneous contacts and absorbed the emanations from the intrusive magma, add their smaller share to the precious-metal production, but are usually richer in the base metals.

Lastly we have a remarkable type of veins, which occur in lava flows near volcanic vents, and which were formed near the surface by hot springs charged with emanations from the molten rocks. These deposits are often wonderfully rich, both in gold and silver. They are the "bonanza" deposits proper; the Comstock, Tonopah, Goldfield, and Cripple Creek are among the more celebrated localities of such veins; few of them are found north of the Canadian boundary and none of them along the main Canadian or American coast, but they are best represented in Nevada, Arizona, Utah and Colorado. In the United States, they yield not less than \$30,000,000 a year.

Going farther south we enter the great mining region of the Mexican plateau. For nearly 400 years an unceasing stream of silver has been poured out of the mines of Mexico, and at the present time the country produces annually about 2,000,000 kg. or 64,000,000 ounces of that metal.² Igneous rocks, both flows and intrusions, abound in Mexico, and practically all of the deposits are of latest Cretaceous or of Tertiary age, thus on the whole more recent than many of those of Canada and the United States.

The most celebrated silver mines are of the type formed in or near volcanic flows near the surface: We need cite only Pachuca, Guanajuato, and Zacatecas; but there are hundreds of other similar districts. Of late the annual gold production has risen sharply to \$20,000,000 or \$25,000,000; part of this comes from silver or base bullion, but the greater part is derived from veins in volcanic rocks similar to those just described and situated at El Oro, in the state of Mexico. It should not be overlooked, however, that there are also in the Cretaceous limestone countless though small intrusive masses of diorite or porphyries around which auriferous or argentiferous veins or contact-metamorphic deposits have formed, and which contribute their share to the production.

The Antilles

Evidences of a feeble mineralization are found in Cuba, Haiti, Porto Rico, and Jamaica and more or less placer gold was obtained, particularly during the 16th Century from the first three islands named. Even now 100 oz. or so are washed annually from the rivers of Porto Rico and

² The total production of silver in Mexico is estimated as 122,500 metric tons, a quantity far greater than that yielded by any other country in the world. (See Bey-schlag, Krusch and Vogt, *Die Lagerstätten*, vol. ii, p. 69 (Stuttgart, 1912).

perhaps the same amount from those of Haiti. The gold seems to be derived from the vicinity of intrusives such as diorite and serpentine, in part, if not altogether, of post-Cretaceous age. The gold placers of Cuba were situated in the middle part of the island, in the Santa Clara and Puerto Principe provinces. Those of Haiti are said to have been highly productive in the early days of the Spanish régime.

Central America

The Cordillera does not continue as an unbroken chain from Mexico into Colombia. The structure of Central America is complex, with short easterly trending ranges of older rocks in Guatemala and Honduras. Farther south these older rocks are submerged beneath Tertiary and recent lavas, in part andesitic. The isthmus connecting the two Americas is in fact marked by a chain of volcanic cones, many of which are active.

Though some mineralization is found in the older rocks of pre-Tertiary age the valuable deposits are mainly in andesitic or rhyolitic rocks, and belong clearly to the class of veins which were formed near the surface. Some of these yield mainly gold, but in many cases they are of the well-known type in which gold and silver occur together without notable amounts of the baser metals. The annual production of Central America ranges from \$1,500,000 to \$4,500,000 in gold and from 50,000 to 75,000 kg. of silver.

Guatemala contributes but little though there are many prospects and placers on Motagua River on the Atlantic side.

Honduras has the reputation of great richness. Its placers of Olancho and Choluteca were worked by the early conquerors. At present the greatest part of its production comes from the gold-silver mine of Rosario near Tegucigalpa. The republic is the largest silver producer in Central America. Gold to the value of about \$600,000 is produced annually in each of the three states, San Salvador, Nicaragua, and Costa Rica. In Nicaragua rich placers have been worked in the Prinzapolca and other Caribbean rivers, and the gold mining district of Pis-pis in the north-eastern part of the republic has lately attracted much attention. Costa Rica has had a considerable production from the placers of Monte Aguacate. The Abengarez and Montezuma lode mines, on the Pacific side, are now the chief producers. In San Salvador the production comes largely from the Butters mines. All of these veins appear to be contained in andesite or rhyolite.

We find the same condition in Panama, though at present there is little production from this state. The Espiritu Santo mine at Cana near the Colombian boundary has been worked from the 17th to the 20th century and the deposit is contained in Tertiary andesite.*

* Malcolm MacLaren, gold, London, 1908.

The South American Cordillera

General Features.—From Cape Horn to Colombia the South American Cordillera or Andes forms a continuous chain closely following the coast. Its width ranges from 100 miles near Magellan Strait to 500 miles in the latitude of Bolivia. North of Bolivia it again contracts to a width of about 300 miles. It is thus, considering its length, a narrow mountain chain, but nevertheless generally made up of three longitudinal units. In the north they are known as the Eastern, Central, and Western Cordillera or by other local names. In Peru they are spoken of as the Coast, Sierra, and Montaña regions, the last being the eastern slope of the Andes. In the south there are locally four subdivisions: the Coast Range, the Western and the Eastern Cordillera, and the pre-Cordilleras or Front Ranges. Between the eastern and western range lies, in Bolivia, the high plateau or "Altiplanicie." In places, as in northern Chile and in Bolivia, the western range itself partakes of the character of a plateau.³

Two ranges stand out by reason of great altitudes, both being rich in mineral deposits. One is the Sierra Blanca of northern Peru, in the Western Cordillera; the other is the Cordillera Real of eastern Bolivia which includes the high summits of Sorata and Illimani. Fig. 2 shows the distribution of the precious-metal deposits in South America.

III. GEOLOGY OF SOUTH AMERICA

Introduction

It will be admitted that it is no easy task to condense in a few pages what is known of the geology of a continent; and for the imperfections and omissions in this account I must therefore ask the indulgence of the reader.

Broadly speaking, the most prominent formations of the Andes are the Cretaceous sediments, which extend almost without interruption from northern Colombia to Tierra del Fuego. Of scarcely less importance, though smaller in area, are the Tertiary and Recent lava flows and the intrusive masses of early Tertiary age. No great intrusions of Cretaceous age appear to exist in South America, although the volcanic activity in the Jurassic and Cretaceous was intense and yielded heavy masses of lava flows intercalated in these formations.

As far as known, the pre-Cambrian is only exposed in the north and on the Argentine side of the Bolivian high plateau.

³ Isaiah Bowman: Physiography of the Central Andes, *American Journal of Science*, 4th Ser., vol. xxviii, pp. 197 and 373 (1899).

Colombia and Ecuador

The work of W. Sievers, A. Hettner, A. Stuebel and Theodore and W. A. Wolf permits a general view of the geology of these countries. As already emphasized, the Andes of Colombia, divided into three chains, do not continue toward the isthmus, but bend eastward toward Venezuela. The coast, both in Colombia and Ecuador, is occupied by Tertiary strata. The Cordilleras consist in general of a core of crystalline schistose rocks which are generally referred to the pre-Cambrian. Above these there is a great break in both countries: The Paleozoic and the early Mesozoic apparently are missing. Instead, the Cretaceous overlies the schists and the extensive beds are divided into the Lower and Upper, the latter being overlain by the Guaderas beds, probably also Cretaceous. There was no marked folding during the Cretaceous. Quartz monzonites and allied rocks are reported from many places; they are older than the Tertiary and younger than the upper Cretaceous. Flows of "Labrador porphyrite" and tuffs are embedded in the Cretaceous.⁴

The Cretaceous is unconformably overlain by the Tertiary. Latites and tuffs represent the volcanic activity of the early Tertiary, continued by the *ejectamenta* of a series of recent volcanoes, most strongly represented in Ecuador.

A sketch map of the general geology of Ecuador, by W. A. Wolf,⁵ shows similar conditions. There is a broad belt of Tertiary beds along the coast adjoined by a narrow belt of Cretaceous with associated eruptives. Then follows the volcanic belt, Quito being placed at its eastern margin, and the main Cordillera east of that city is built of granite and crystalline schists, all probably pre-Cambrian.

Peru

Much information on the geology of Peru is contained in the publications of the Cuerpo de Ingenieros de Minas at Lima, which include also some of the important writings of Prof. G. Steinmann. The results refer mainly to the western and central Cordillera, and the geological features of the Montaña slope, clad in tropical vegetation, are as yet little elucidated.

Steinmann's profiles⁶ from the Pacific to Rio Marañon show 180 km.

⁴ E. Lehmann: Beiträge zur Petrographie des Gebietes am oberen Rio Magdalena, *Tschermak's Mineralogische u. Petrographische Mitteilungen*, vol. xxx, pp. 233 to 280 (1911).

⁵ Sketch of the Geology of Ecuador, condensed in *Mining and Scientific Press*, vol. cv, No. 4 (July 27, 1912).

⁶ Gebirgsbildung und Massengesteine in der Kordillere Südamerikas, *Geologische Rundschau*, vol. i, Fas. 1-3, 1910.

Ueber gebundene Erzgänge in der Kordillere Südamerikas, International Mining Congress, Düsseldorf, 1910.

of upper and lower Cretaceous beds with interbedded volcanics, strongly folded and in part overturned toward the east. There are in these Cretaceous rocks numerous early Tertiary intrusions of granodiorite and porphyries ("*Andesitische Tiefengesteine*"), but few of them are more than 10 km. in width. In the valley of the Marañon, old ("pre-Devonian") schists and granites appear for the first time and probably form the continuation of the pre-Cambrian of Colombia and Ecuador. The porphyritic intrusions are extremely numerous, and Steinmann refers to them as "laccoliths" though usually they have a vertical attitude, conformable to the surrounding sediments. Farther south the granodioritic batholiths become even more abundant, one exposed in the Rimac River being 50 km. in width. They always metamorphose the surrounding Cretaceous limestone.

Bolivia and Southern Peru

A section across this region, recently described by J. A. Douglas,⁷ is 330 km. long but does not include the whole of the Montaña slope. Here the Andes are divided into the Western Cordillera, the Bolivian High Plateau or the "Altiplanicie" and the Eastern Cordillera or the Cordillera Real. The latter includes the highest summits, Illimani and Sorata, but contains no volcanoes or large masses of volcanic rocks. It is largely built of older Paleozoic sediments (Cambrian, Ordovician, Silurian and Devonian), mostly slates and sandstones, and these are intruded by masses of granite, diorite and porphyries. The upper Devonian and the lower Carboniferous are both absent.

In the "Altiplanicie" we find the same folded Paleozoics, with transgressing Cretaceous in part terrigenous sediments, such as those of Corocoro. The Cretaceous is covered by post-Miocene andesites.

The Western Cordillera along this section is essentially a volcanic range with numerous dormant or extinct volcanoes, and vast accumulations of lavas including rhyolite, trachyte and andesite.

Underlying these rocks and beautifully exposed along the Chilean coast as far north as Arica, are Jurassic and Cretaceous strata interbedded with contemporary lavas, and intruded by early Tertiary granular rock. The latter range from quartz monzonite to quartz diorites, and are accompanied by pegmatite dikes, many of which carry tourmaline. These intrusives are best exposed in the cañons.

Mr. Douglas regards the intrusive rocks of the Eastern Cordillera as post-Devonian and pre-Jurassic in age; but this is apparently not proved, some authors calling them early Tertiary.

⁷ Section across the Andes in Peru and Bolivia, *Quarterly Journal of Science*, vol. lxx, pt. I, pp. 1 to 53 (1914).

Chile

Conditions similar to those just described prevail in Chile. We find here, however, a coast-range of lower elevations, largely made up of Mesozoic sediments with interbedded volcanics and a main western range, plateau-like in the north, which is surmounted by a long line of active volcanoes and often really constitutes the western margin of the Altiplanicie. The basement on which the volcanic cones rest is largely of Mesozoic sediments, more or less abundantly intruded by granodioritic rocks. South of Concepcion the intrusive granitic rocks increase in volume and are bordered on the west by metamorphosed sediments of doubtful age in Chiloe island and the Taytao peninsula. Quensel's³ researches have shown that a vast body of quartz dioritic intrusive, similar to the batholith of British Columbia, but of greater length, follows the coast from Puerto Montt down to the extreme tip of the continent.

On the east side this batholith is almost continuously adjoined by Mesozoic sediments⁴ in which great flows of "quartz porphyry," "porphyrites" and their tuffs are embedded. These continue for 1,000 miles or more northward along the eastern slopes. The nomenclature is open to objection; the rocks are rather rhyolites, andesites, etc.

In this Patagonian region the distinction between the coast, central and east Cordillera is less clearly marked. Pre-Cordilleras or front ranges appear on the east side and are made up of granitic laccolithic intrusions. East of these, again, are found vast table-lands of basalt and other Tertiary effusives, which slope eastward and in places reach almost across Patagonia.

In southern Patagonia there is only one period of folding, involving Cretaceous and Tertiary beds, while farther north and indeed through the whole chain of the Andes there are two periods of folding, one Jurassic or older, the other late Mesozoic or early Tertiary.

Argentina

The recent work of Argentine geologists, such as R. Stappenbeck, H. Keidel, and others, has given us a clear idea of conditions along the eastern slope of the Andes. This is rarely a simple slope but usually a succession of ridges, the more easterly of which are called the pre-Cordilleras. In the extreme north, in Salta and Jujuy provinces, really the

³ Geologisch-petrographische Studien in der Patagonischen Cordillera (Upsala, 1911).

⁴ Practically all of the sediments of the region of Magellan Straits and Tierra del Fuego are considered as belonging to the Mesozoic series. On the west coast the batholithic rocks face the sea.

continuation of the Bolivian Altiplanicie, the Paleozoic rests according to Keidel¹⁰ with marked discordance on phyllites and quartzites of probably pre-Cambrian age.

In the eastern main Cordillera the marine Mesozoic (Jurassic and Cretaceous) rests unconformably on the basement of Paleozoic slates and includes great masses of flows of "quartz porphyries" and "melaphyres," i.e., rhyolites and basalts.

In the pre-Cordillera of San Juan and Mendoza¹¹ there are heavy continental deposits of upper Carboniferous to upper Triassic age, resting on a Paleozoic folded basement. According to I. Bowman and other geologists these "pre-Cordilleras" continue northward into Bolivia and here also consist, in large part, of continental sandstone deposits. Small areas of porphyries and granite are intruded in these rocks. The series is gently folded toward the east.

On the eastern slopes of the Andes, sedimentary rocks generally predominate. Two periods of folding are recognized: an older Paleozoic and a younger Tertiary movement, the latter being designated as the properly Andene disturbances. Along the eastern border the latter is marked by overthrusts and overturned folds.

IV. DISTRIBUTION OF SOUTH AMERICAN DEPOSITS OF GOLD AND SILVER

In the following paragraphs a brief summary is given of the distribution of the precious metal deposits in each of the cordilleran states of South America.

Colombia

In Colombia we find the principal gold belt of the Andes, which under adverse circumstances yields annually a notable production of \$3,000,000 to \$4,000,000. This production is probably capable of considerable expansion.¹² The total yield of that country, as calculated by Vincente Restrepo, amounts to about \$700,000,000; therefore Colombia takes its place among the great gold producing regions of the world.

The deposits are mainly in the western and central ranges, which do not continue northward into Panama, but bend eastward toward Vene-

¹⁰ Ueber den Ban der Argentinischen Anden. Sitzungsberichte der Kaiserlichen-Königlichen Akademie der Wissenschaften (Wien, 1907).

Die neueren Ergebnisse der Staatlichen geologischen Untersuchungen in Argentinien. *Compte Rendu*, 11th Session Congrès Géologique International, Stockholm, pp. 1127 to 1141 (1910).

¹¹ R. Stappenbeck: La Pre-Cordillera de San Juan y Mendoza, *Anales*, Ministerio de Agricultura, Sección geológica, Tomo iv, No. 3 (Buenos Aires).

¹² The latest statistics for 1914 show a very marked increase in the production of Colombia, the figure being \$4,678,600.

zuela. The eastern range, in which the city of Bogota is situated, appears to be lacking in precious-metal deposits.

Heavy gravel deposits containing gold and platinum are found along the coast on the Atrato and San Juan rivers, but the richest placers, some of which are now being dredged successfully, lie along the drainage trending northward, in the Magdalena, Porce, Cauca, and Nechi rivers. These are of great value deposits though difficulties of transportation and climate have interfered with their successful exploitation.

The majority of the lode mines are in the departments of Antioquia, Cauca, Bolivar, Tolima, and Santander, of which the first two are the most important.

The deposits are mostly typical quartz veins, often with crystallized native gold, and more or less pyrite, pyrrhotite, arsenopyrite, chalcopyrite, galena and blende, occasionally also tellurides. They are closely related to the California type and undoubtedly allied in their genesis to intrusive rocks. Though the deposits usually occur in granite and schists of probable pre-Cambrian age, porphyries or monzonites of much later date (probably early Tertiary) are usually found close to them. These intrusive rocks have sometimes been described as andesites or rhyolites.¹³

Among the deposits there is also another class, the representatives of which yield gold and silver or silver alone, and which occur in undoubted flow rocks, such as andesite and rhyolite. Many of them contain stibnite, tetrahedrite, pyrargyrite, jamesonite and stephanite and are formed under materially different conditions and near the surface. Such mines are those at Marmato and Echandia in Cauca, and those near Manizales on the boundary of Tolima and Antioquia.

Altogether Colombia must be considered as the most promising gold-bearing region of South America.

Ecuador

Apparently Ecuador is not rich in deposits of precious metals. The coast is occupied by Cretaceous and Tertiary sediments, the former including some intrusive rocks. These are adjoined by a zone of igneous flow rocks of Tertiary or recent age, surmounted by volcanic cones, while, according to W. A. Wolf, the best authority on the subject, the main or Eastern Cordillera is built of ancient schists and crystalline rocks.

Almost the whole of the moderate production of a few hundred thousand dollars comes from the ancient mines at Zaruma near the Peruvian boundary and 50 miles from the coast. According to J. R. Finlay, these veins are contained in a fine-grained diorite. In the Esmer-

¹³ H. W. Nicholas and O. C. Farrington: The Ores of Colombia, *Bulletin*, No. 33, *Field Colombian Museum*, 1896.

aldas near the coast and the Colombian boundary there are placer deposits which have not so far been successfully worked; the eastern ranges are also said to contain placers which may be derived from deposits of pre-Cambrian age.

Peru

The conditions in Peru are very different from those in Colombia. There are relatively few gold deposits—some veins are being worked, and a certain amount of placer gold is obtained from the Montaña region of southern Peru. The annual production of gold is rarely over \$500,000; thanks to the careful work of the Cuerpo de Ingenieros de Minas it is possible to gain an exact idea as to its derivation. Half of the production comes from the copper of Cerro de Pasco. One-sixth is derived from placers and one-fourth from gold-quartz mines proper.

On the other hand, Peru is the leading silver-producing country in South America, the present annual output being about 9,600,000 oz. or 300,000 kg. Of this again more than one-half is derived from the copper mines of Cerro de Pasco, a small amount from lead bullion, and the remainder from silver or gold-silver deposits.

It is well known that Peru has yielded an enormous amount of silver. Professor Vogt has estimated 35,000,000 kg. as the production from 1533 to 1910. Whether this is accurate or not, it is certain that Cerro de Pasco has contributed the greater part of the silver of Peru.

The silver districts are very numerous and generally situated in the Western Cordillera in the departments of Cajamarca, Libertad, Ancachs, Huanuco, Junin (Cerro de Pasco), Lima, Huancavelica, and Arequipa. It would seem that the silver production could be considerably increased.

Geologically there is also a great difference from conditions in Colombia. In Peru and Chile we find along the coast and Central Cordilleras a strong development of Jurassic and particularly Cretaceous sediments, folded and in part overturned toward the east. These Mesozoic sediments contain embedded lava flows of the same age, which, however, do not appear to be of importance as regards mineralization.

According to Prof. G. Steinmann,¹⁴ the great majority of Peruvian deposits are undoubtedly in close genetic connection with numberless small intrusive masses of "andesite," "dacite," or "liparite." These names are confusing for the rocks are really deep-seated dioritic or monzonitic porphyries not at all connected with the "effusives" or flow rocks.

It is thus clear that practically all of the Peruvian deposits are of the intermediate type, formed far below the surface. It is doubtful whether

¹⁴ Gebirgsbildung und Massengesteine in der Kordillere Südamerikas, *Geologische Rundschau*, Bd. 1, Heft 1-3 (1910).

there are in Peru any deposits of the type of the Tonopah, Comstock, or Pachuca veins.

The great Cerro de Pasco deposits, for instance, occur in or close to a stock of "dacite" or "biotite andesite," which has metamorphosed by contact with the surrounding Cretaceous sediments. The proper name would seem to be biotite-diorite porphyry. In their upper levels the veins carried probably secondary silver ores of wonderful richness, while in depth they have been found to contain low-grade copper ores, which now form the basis of a great industrial enterprise.

Besides the smaller bodies of intrusive porphyries, there are also numerous large intrusive masses or "batholiths" of granodioritic rocks. Some of these form the central parts of the great ranges, and they may continue for a long distance with a width sometimes reaching 50 miles. Around these also there has been more or less mineralization, but of a more feeble character than attended the intrusion of the porphyries. The time of intrusion is taken to be early Tertiary.

In the gold-bearing region of southeastern Peru (northeast and north of Lake Titicaca), we find different conditions. Here the folded sedimentary rocks are of early Paleozoic age and more or less intruded by porphyries and granodiorites. This is in the regions of Carabaya and Sandia, and the Inambari basin on the Montaña slope. A very widespread, though not intense, mineralization has taken place; the primary gold deposits are apparently poor but the placers are widely distributed and numerous; partly successful attempts have been made to mine them. This belt is, in fact, the northern continuation of the great tin-silver-gold belt of the eastern range of Bolivia.

Bolivia

Bolivia produces little gold at the present time, but its placers on the Montaña side have at times yielded heavily. They lie on the eastern slope of the great range, east of Lake Titicaca, which counts among its peaks Sorata and Illimani, over each 21,000 ft. in elevation. Celebrated among these were the placers of Tipuani on the east slopes of Sorata, which have yielded great amounts of gold since the time of the conquerors. There are many other localities south of this. Other placers have been worked recently on the San Juan River near the Argentine boundary. At the present time only two gold veins are worked, both in the eastern range and said to be of the "saddle reef" type inclosed in slates and sandstones.¹⁵ The quartz and free gold are accompanied by pyrrhotite, arsenopyrite and pyrite. They thus belong to the intermediate type accompanying intrusive rocks. The ore is of low grade.

¹⁵ F. C. Lincoln: Incaoro Mine, *Mining and Scientific Press*, vol. cviii, No. 14, p. 561 (Apr. 4, 1914).

Bolivia points with pride to its production of silver. The yield from 1553 to 1910 is stated to have been 48,800,000 kg., to which the mines of Potosí are said to have contributed no less than 30,000,000 kg making this district the greatest silver mine the world has known. Nor is this large production entirely a matter of the distant past, for it is said that the *Compagnie de Huanchaca de Bolivia* sent to the markets of the world from its mines, which lie to the south of Potosí, silver and lead to the value of \$50,000,000 between the years 1873 and 1888. At present Bolivia yields 80,000 to 150,000 kg. (2,500,000 to 4,800,000 fine oz.) per annum. A large part of this comes as a byproduct from the tin mines; another part is derived from the mines near Huanchaca.

The great mineral belt of Bolivia lies in the extremely rough chain which forms the eastern border of the "Altiplanicie" or high plateaus of that country, a region of Paleozoic folded slates with intrusive cores of diorite, granite and porphyritic intrusions. Volcanoes and lava flows are generally absent. In this range there has been produced a widespread mineralization, in part of gold but more characteristically of the peculiar type of Bolivian tin veins first described by Stelzner, and carrying both silver and tin. All these deposits extending from the Peruvian boundary almost to the Argentina border are certainly of the deep-seated type connected with intrusive rocks. In general, these are porphyritic and may be designated as quartz porphyry or granitic porphyry. In the literature they are frequently referred to as andesite and rhyolite, which usage tends to produce an erroneous impression. There are probably no deposits in Bolivia of the type formed near the surface in flow rocks.

Interesting changes are observed in depth. Just as the Cerro de Pasco silver veins turned into low-grade copper veins in depth, so the wonderfully rich silver veins of Potosí are shown, as the great mountain is penetrated by deep adits, to have been transformed into pyritic tin-bearing veins. The silver production from this district is now of smaller moment than formerly.

Chile

Lack of statistical data makes it difficult to review at a distance the deposits of Chile. The republic of Chile, so progressive in other respects, has made little effort to study or keep account of its mineral deposits.

The narrow strip of coast occupied by the republic is in few places more than 150 miles wide, but extends from the 18th to the 56th degrees of south latitude. From latitude 20° to 36°, a distance of 1,200 miles, this part of the Pacific slope is mineralized in a complex and manifold way, while the remaining distance to Cape Horn contains extremely few gold and silver deposits. This is surely a remarkable feature.

It is not my purpose to describe the great resources in copper which have lately been developed in Chile; these deposits as a rule contain little or nothing of the precious metals. Chile has never yielded very large amounts of gold. At the present time the production appears to be diminishing, as may be seen from Table I, and does not exceed a few hundred thousand dollars per annum. The silver production is a little more valuable, but scarcely reaches 30,000 kg. (960,000 oz.) per annum. At no time has the silver reached the figures of Bolivia and Peru, although the rich deposits of the northern coast during a short period in the 19th century made Chile prominent among silver-producing countries.

The present moribund condition of the industry certainly appears strange when we consider the almost continuous chain of mining districts extending over a distance of 1,200 miles.

The total gold production of Chile from the 16th century up to 1906 is estimated by Herrman ¹⁶ at \$212,000,000, or less than a third of that of Colombia.

The total production of silver is estimated at 6,600,000 kg., only a small part, it will be observed, of the yield of Peru and Bolivia.

The northern half of Chile shows in general a geological structure similar to that of the Western Cordillera of Peru. The Jurassic and Cretaceous formations are strongly developed with contemporaneous lava flows of great volume. Into these are intruded granite porphyries and diorite porphyries in smaller stocks, as well as many batholithic masses of granodioritic rocks. Both of these kinds of intrusions have brought mineral deposits. There are finally heavy masses of late Tertiary lava flows, and in these we find a few representatives of the type of precious-metal veins which were formed near the surface. The great majority of deposits are associated with intrusive rocks and many of these carry tourmaline with copper and gold, indicating that they were formed under conditions of high temperature. It is necessary to read the descriptions critically, for here, as elsewhere in South American literature, andesite, dacite and rhyolite are names often used for intrusive Tertiary rocks, a survival of the old view that any Tertiary volcanic rock must belong to one of these rock types.

Some gold-bearing veins are found in rhyolite and allied flow rocks, for instance, at Guanaco, southeast of Antofagasta, probably also at Sierra Overa, southeast of Taltal, and at Andacollo, southwest of Coquimbo. Other veins carrying both silver and gold occur, according to Moericke,¹⁷ in andesite flows, in part tuffaceous, for instance, at Batuco

¹⁶ *La Produccion en Chile de los Metales y Minerales Santiago de Chile* (1903).

See also Malcolm MacLaren: *Gold*, p. 662 (London, 1908).

¹⁷ W. Moericke: *Einige Beobachtungen ueber Chilenische Erzlagerstätten Tschermaks Min. u. Pet. Mitteilungen*, vol. xii, pp. 186 to 198 (1891).

and Cerro Blanco. According to Moericke all veins of this type seem to have a tendency to play out at a depth of a few hundred feet.

Much more numerous are the gold quartz veins connected with intrusives, such as granites and quartz diorites. We find them at Canutillo,¹⁸ north of Taltal, in diorite intrusive in Cretaceous limestone. Others are found associated with tourmaline and copper ores at Remolinos in Atacama, at Tamaya and La Higuera in Coquimbo, and at Las Condes in Santiago. Another gold belt extends from Coquimbo down to Santiago, and to Roncagua and Talca, south of this city.¹⁹ These quartz veins occur mostly in granite near the contact of schist.

While silver is sometimes associated with gold, the richest silver mines of Chile, which yielded great amounts of the metal in the 19th century, occur as a rule separate in Mesozoic limestone, intruded by or interbedded with greenstones of various kinds. They are characterized by extremely rich ore and antimonial and arsenical silver minerals; some of them also contain silver amalgam. Their genesis is doubtful. The gangue is mainly calcite. In depth these veins also are disappointing and the silver production of Chile is now only a fraction of what it was when these mines were in bonanza.

Among these celebrated districts, mainly situated along the coast, are Huantajaya and Challacollo near Iquique, Chanarcillo (50 miles south of Copiapó), and finally a group of districts including Arqueros and Condoriaco (100 miles south of Copiapó).

The great low-grade copper deposits, such as Braden and Chuquicamata, appear to contain very little of the precious metals.

In remarkable contrast to the northern half, so rich in precious-metal deposits, the southern part of the republic appears to be amazingly poor in mineral deposits. Scarcely any mines are reported from this region except an auriferous vein worked by the Spaniards near Valdivia, and some auriferous beach sands along the coast, for instance, on Chiloe island. Not until we reach the Straits of Magellan are there any producing deposits. At Punta Arenas on these straits and on the eastern side of the Andes there are gold-bearing gravels rich enough to justify dredging. Similar placers are found on the south side of the Straits in Tierra del Fuego. About 1902 a dozen dredges were erected here and for a number of years these gravels have contributed largely to the gold production of Chile, yielding annually up to \$100,000. The production has decreased materially during the last few years, owing it is said, to difficulties in dredging the bouldery deposits.

¹⁸ S. H. Loram: Notes on the Gold District of Canutillo, Chile, *Trans.*, vol. xxxv, p. 696 (1906).

¹⁹ E. D. Pope: Gold Mining in Chile, *The Mining Magazine (London)*, vol. xiii, No. 1, pp. 33 to 36 (July, 1915).

The difference in mineralization is intimately connected with a great change in topographical and geological conditions.²⁰ From latitude 42° down to Cape Horn the Cordillera is invaded by the ocean and by ice. Its westerly margin is cut up into an intricate system of fjords and its summits are clad in the armor of immense ice fields. A huge batholith of granitic and dioritic rocks occupies the whole western range, probably from Puerto Montt to the tip of the continent. This constitutes a striking analogue to the batholith of British Colombia; it is of greater length and its width in many places reaches 100 km. On the east side the ice fields often cover its margins. On the west side the adjoining sedimentary rocks are largely submerged, but on Wellington and Chiloe islands these western sedimentaries begin to appear as metamorphosed schists of uncertain age. All along the eastern side the batholith is intruded in Mesozoic (Cretaceous and Jurassic) rocks. Along the eastern edge of the latter we find again front ranges of granitic laccoliths, such as Cerro Payne, Cerro Balmaceda, etc., most of them consisting of granitic rocks. There is little doubt that the gold placers of Punta Arenas have derived their metal from the mineralization along the eastern side of the great Chilean batholith. It would be strange if this batholith would not be accompanied by mineral deposits. That no such have been found may in part be accounted for by the extensive present and former glaciation which would destroy most placer deposits and to the fact that the region is extremely inhospitable. It would not be surprising if scientific prospecting along the borders of this batholith should lead to the discovery of gold-bearing deposits.

Argentina

The present Argentine production of gold and silver is very small indeed, and the country has never yielded large amounts of these metals.

The Sierras of the Pampas, like that extending from San Luis to Cordova, contain a feeble pre-Cambrian or early Cambrian mineralization, referred to above, but these quartz veins appear to be poor in gold and silver. In the same vicinity there is also evidence of a much later development of gold deposits, perhaps connected with the effusion of Tertiary andesitic lavas, but these veins which have the character of crushed or sheeted zones are also poor in gold.²¹

The whole eastern slope of the Andes from the Bolivian plateau to the latitude of Santiago de Chile shows a relatively feeble mineralization.

²⁰ P. D. Quensel: Geologisch-petrographische Studien in der Patagonischen Cordillera (Upsala, 1911).

²¹ E. Gerth: Constitucion geológica de la Provincia de San Luis, *Anales Ministerio de Agricultura, Sección Geológico*, Tomo x, No. 2, 1914.

The slopes of the central Cordillera and the pre-Cordilleras are largely composed of sedimentary rocks folded, overturned, and overthrust toward the east, with relatively small and inconspicuous areas of igneous rocks,²² which are designated as andesites and dacites, but which in reality seem to be holocrystalline intrusives. There are also smaller areas of granular rocks of Tertiary age, which were designated as "Anden diorite" by Stelzner.

Gold, silver, and copper prospects are rather abundant, but at very few places has serious work been undertaken. The most important deposit, located at Famatina is a copper-bearing vein with sulpharsenides and antimonides of copper and very little gold and silver.

The eastern slope of the Andes in the northern half of the Argentine Republic is comparable in a way to the eastern Rocky Mountain chain of Canada. Both show overturned folds and overthrusts toward the east, with comparatively little of intrusive rocks and attendant mineralization.

The gold-silver-tin belt of the Bolivian eastern Cordillera apparently does not enter the Argentine territory.

No lode deposits are reported south of Mendoza, except on the head waters of Neuquen River, at about the latitude of Concepcion in Chile, where there is a mining district of gold-bearing quartz veins in granite of uncertain age. Considerable work has been done on these, but the expected production does not seem to have been realized. The ore is apparently of low grade. The only other precious-metal deposits reported from the eastern slope of the Andes in Patagonia are placers of doubtful value on the headwaters of Chubut, Rio Gallegos and other streams. Placers and some lode mines have been taken up at various places on the Argentine Tierra del Fuego, but little information is available as to their values.

As observed above, the Mesozoic beds of the Patagonian Cordillera and eastern Cordillera are intruded by laccolithic and batholithic masses of granitic rocks, and careful prospecting might well yield favorable results. The glaciation probably would have destroyed any placers which may have existed in this region, and this guide for the prospector is, therefore, generally lacking.

V. COMPARISON OF THE TWO CONTINENTS

It has been shown that the pre-Cambrian and early Paleozoic gold deposits predominate in the eastern part of North and South America; that they are scattered irregularly over a wide territory and do not form well-defined belts except locally; and that the heavy production is very much localized. There is reason to believe that such deposits occur here

²² R. Stappenbeck: *La Pre-Cordillera de San Juan y Mendoza*, *Ibid.*, Tomo iv, No. 3.

and there in the pre-Cambrian rocks of the Cordilleran regions, though they are not easily differentiated from the later Cordilleran period of mineralization. We note the marked localization of rich deposits in the Black Hills and in the Porcupine, which may be compared to the strongly auriferous districts of the Guianas and Minas Geraes. We observe, also, that as far as this earliest mineralization is concerned, both continents are about equally rich. No silver deposits of this period, such as are concentrated to such a remarkable degree at Cobalt, Ont., are known from South America.

In the Cordilleran region of South America the principal and almost the only period of mineralization seems to be that of the early Tertiary, while in North America an important series of deposits dates from the early Cretaceous. The batholithic and smaller intrusions in South America all appear to date from early Tertiary, and the evidence of close connection between intrusion and mineralization is cumulatively strong and convincing. The same general principles of association of the two agencies apply in the two continents.

So far, no definite evidence has been adduced that the great lava flows of the Jurassic and Cretaceous contain mineral deposits of that general age. In North America many intrusions—in fact the greatest batholiths—date from the earliest Cretaceous. No such occurrences are found in South America.

From northern Mexico to Chile the Cretaceous is by far the most prominent of the sedimentary formations, while the Carboniferous limestone, so important for the mineralization of the Cordilleras in the United States, is entirely lacking.

Another interesting feature is the great scarcity in South America of Tertiary deposits of gold and silver occurring in late Tertiary lavas and formed close to the surface. Popularly the majority of deposits in South America are ascribed to this group, and even the latest text-books fall into this error. There are some of these interesting and rich deposits in the southern provinces of Colombia, but none have been recorded in Peru and Bolivia. In Chile they reappear at some places such as Guanaco, Batuco, and Cerro Blanco,²³ but compared to the deposits of other classes they are rare. This is remarkable, when we consider the widespread occurrence in Central America, Mexico, and the Western United States of deposits of the type of Pachuca, Guanajuato, the Comstock, and Tonopah, all marked by certain well-defined characteristics.

A large number of deposits in Colombia, Bolivia, and Chile approach the high-temperature veins by their content of pyrrhotite and tourmaline.

A curious fact is that, so far, no contact-metamorphic deposits are described from Peru and Chile, although metamorphism of the Cretaceous

²³ The copper deposit at the Braden mine near Santiago also appears to belong in this class.

limestone by the granodioritic intrusions is often mentioned. In the Cordillera Real of Bolivia they would hardly be expected, for there the intruded rock is generally a slate or sandstone.

The poverty of the eastern front ranges of the Andes is paralleled by the lack of precious metal deposits in the eastern or Rocky Mountain range of Canada.

North America stands out in its richness of placer deposits derived from veins of the Cretaceous intrusive period. In South America there is no real counterpart to the great placers of California, Idaho, Montana, Alaska, and Yukon Territory.

The placer deposits of the Andes, which were locally rich, were mostly found on the eastern slopes of the eastern ranges, and were derived from gold-bearing veins in Paleozoic slates, with intruded granite porphyries and allied rocks.

Colombia stands out prominently as the most valuable gold-bearing region of the Andes, from which, in spite of many difficulties, we may expect a considerably increased production.

The next region is formed by Peru and northern Chile—a region of very numerous mining districts in which the mineralization is chiefly in the direction of silver and copper with a few gold-bearing localities, which, however, do not seem to be able to achieve great production. No doubt the silver output could be materially increased, particularly where silver occurs with copper. In looking over the numerous gold-bearing districts of central Chile, the student would like to ascertain the conditions which in so favored a country have held back the production to such a marked degree.

The third region is formed by the Cordillera Real of Bolivia with its rich mineralization of tin, silver, and (subordinately) gold. Undoubtedly this region is one of the most promising in South America.

Lastly a striking contrast is presented between the two tips of the Cordilleran chain. At the north is Alaska, rich in gold, at the south is the Patagonian Cordillera, with its gigantic batholith, so promising theoretically, so barren in reality. It is barely possible that theory may be vindicated and that valuable deposits may be found hereafter in this vicinity.

It is difficult to avoid the conclusion that the South American Andes are somewhat less intensely mineralized in precious metals than the corresponding chain in the northern continent, and that even progress and enterprise will be unable to raise its production of gold and silver to approach the figures attained by North America.

APPENDIX

PRODUCTION OF SILVER IN THE AMERICAN CONTINENT FOR 1913
(From the Reports of the Director of the Mint and from tables in *Mineral Industry*)

(Value of 1 kg., \$19)

South America

	Kilograms	Value
Colombia.....	42,100	\$ 800,000
Peru.....	299,132	5,683,500
Bolivia.....	81,300	1,544,700
Chile.....	30,178 ¹	573,400
Argentina.....	1,097	20,800
	453,807	\$8,622,400

Central America

Kilograms	Value
66,427 ²	\$1,262,100

North America

	Kilograms	Value
Canada.....	990,500	\$18,984,000
United States.....	2,074,700	40,348,100
Mexico.....	2,112,400 ³	40,144,300
	5,177,600	\$99,476,400

¹ Figures of 1912.² For 1912: See Report of Director of the Mint for Calendar year 1913, p. 247.³ Fiscal year 1912-1913.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the Arizona meeting, September, 1916, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1916. Any discussion offered thereafter should preferably be in the form of a new paper.

Principles of Natural-Gas Leasehold Valuation

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(Arizona Meeting, September, 1916)

Magnitude and Economic Importance

THE magnitude and economic importance of the problem of correctly valuing natural-gas leaseholds become evident when we consider that:

(a) Natural gas is handled in 55 per cent. of the gas distributing plants in the United States.

(b) Present known natural-gas acreage forms 47 per cent. of the total known mineral-land acreage in the United States.

(c) Five acres of land are now required to protect and maintain continuous service to each of the 2,000,000 domestic natural-gas consumers in the United States.

(d) The cost of acquiring and maintaining this acreage of an expendible resource represents a substantial part of the cost of the natural-gas service to the consumer.

(e) "The right of a citizen by means of his ownership of or his mining leases on land to draw gas from beneath its surface is property and sometimes valuable property."¹

A large number of other court decisions have established the basic legal principle that the right to drill for gas within a given area constitutes an interest in the land itself, and is necessarily an exclusive property right.

Definition of Vested Interests or Rights

"Vested interests are economic interests which are legally recognized to be such that they cannot be impaired by public action, directly or indirectly, without indemnification. Vested interests are largely property interests. The recognition of an interest as a vested interest gives it some of the attributes of property, and by American courts it would be comprised in their very inclusive concept of property. Otherwise than through property, vested interests generally arise through contract."²

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¹ U. S. Circuit Court, *Haskell vs. Cowham*, 187 Fed. Rep., p. 403.

² Richard T. Ely: *Property and Contract in Their Relations to the Distribution of Wealth*.

Definition of Lease

A natural-gas lease is a contract for a consideration establishing a vested right to enter upon a definitely described parcel of land, for a determined period, to prospect for, reduce to possession, remove and market natural gas. The vested right is the crux of the whole matter, and it is immaterial whether the instrument creating it is called a "lease," "contract," "grant," or "deed of conveyance."

"Oil and gas leases, generally speaking, are not strictly leases as defined in the law of landlord and tenant. They are in the nature of written licenses, with a conditional grant conveying the grantor's interest in the gas or oil well, providing that gas and oil is found in paying quantities. It is well settled that the title of the lessee is inchoate until such discovery, at which point of time he acquires a vested estate in the mineral itself."³

"In a given tract of land it is always a matter of doubt to what extent, if any, mineral may exist in paying quantities, until very considerable development work has been performed, which requires in most instances large expenditure of capital. For this and other reasons, a custom long ago arose for the owner of supposed mineral land to grant to a mine operator the right to enter upon the land and search for and extract mineral, and the form which the contracting parties pretty generally adopted to express their agreement was a 'lease,' which purported to entitle the 'lessee' to occupy such part of the premises as was necessary to carry on his mining operations, and to use, mine and extract the minerals therefrom."⁴

License and Lease Distinguished

"A license is an authority to go upon the land of the licensor to do an act or series of acts there, but passes no estate or interest in the land. It is technically an authority to do something on the land of another without passing an estate in the land. The distinction between lease and license is that the former is a distinct conveyance of an actual interest or estate in the lands, while the latter confers a mere incorporeal right to be exercised in the lands of others."⁵

Definition of Property

Property in the legal and economic sense means an absolute, entire and exclusive right to do anything, and embraces material things, labor or services, and rights. It is important to bear in mind that the term means not only the thing owned but also every right which accompanies such ownership and is its incident. Thus, the right to drill at some future time a gas well on a certain parcel of land is property in the same sense that the land itself is property. A long line of court decisions⁶ has definitely established what the term property includes.

³ Lindley on Mines, 3d ed., p. 2144.

⁴ *Idem*, p. 2134.

⁵ *Idem*, pp. 2129 to 2130.

⁶ A list of cases is given in Ely's *Property and Contract in Their Relations to the Distribution of Wealth*, p. 855.

Definition of Science of Valuation

The science of valuation is an interdependent mixture of law, economics, and engineering: Law, to establish the governing ethical principles; economics, to establish the criteria of value or monetary standards; engineering, to apply the general criteria of value to any given specific case in accordance with ethical principles.

Definition of Value

Value is the relation of two services, and implies two things—utility and scarcity. The idea of value entered into the world for the first time when a man said to his brother, "Do this for me, and I will do that for you." They had come to an agreement. Then we could say the two services were worth each other. The word "value" means the exchange power which one commodity or service has in relation to another. Value may also be defined as the price of a given unity of wealth multiplied by the quantity or number of units, and hence is an expression of the relative desirability of different classes of wealth or property in terms of a common standard.

Market Value

This is an adjustment of two services as between a willing seller and a willing buyer under open conditions of competition, without any coercion from either party. The sale value of property is indicated by recent prices for similar property in the same locality. Sale values are not uniform for the same kind and quality of goods, even in the same locality, unless there is a complete general knowledge of current prices and the desire to buy is approximately equal to the pressure for sale.

Price

A price, when expressed in terms of a common standard of value, is the amount of money accepted in exchange for a unit of goods. Prices resulting from exchanges are agreements between two persons as to the value of a unit of property. Prices are, therefore, of human and mental origin, and are established by economic causes which influence the opinions of the majority of purchasers and sellers.

Wealth and Property

"Wealth is defined as material objects owned by human beings. Property is the right represented by ownership, and is a material object itself. The benefits derived from wealth make its ownership desirable, and the right to enjoy these benefits is indicated by the term 'property.'"¹

¹ H. H. Chapman: *Forest Valuation*.

Gas is a Mineral

Although natural gas is a mineral, it cannot be measured in terms of acres, like coal, ore, or solid minerals. On account of its fugitive and wandering nature it develops characteristics which differentiate natural gas valuation problems from all other mineral valuation questions. It is important to note that it is not a continuous process, but a fixed storage reservoir that must be considered, which storage will be depleted, without regeneration, as the supply is removed, and that the transportation from the mining district to the ultimate consumer can take place only through pipe lines.

Transient Nature of Mineral Values

"In the administration of natural resources, we must bear in mind some important differences. Mineral rights and surface rights bear quite different relations to society. Surface rights have a permanent, underground rights but a transient value. The more intensely a farmer cultivates his land, the more valuable does it become to him and to the community. If development takes the shape of permanent and useful structures on the land, the increase in value to the individual and to the state may reach vast proportions. Indeed, it is not possible to anticipate the maximum usefulness which surface rights may acquire; but we know that when these rights are being used to the best advantage the country is also reaping the greatest benefit, and it is then that the life of the right appears longest and most useful.

"A mineral right is of a different nature. The more it is worked, the smaller becomes its intrinsic value. It is of value to the community only at the time, and in proportion to the extent of its productiveness; but the greater this temporary usefulness, the shorter its duration.

"The agricultural claimant locates a 'homestead,' while the prospector locates a 'claim.' The former indicates permanency of tenure, while the latter suggests a temporary possession."⁸

Judicial Recognition of Hazard

"We take judicial notice of the fact that mining for oil and gas is a very hazardous and dangerous business, involving great risk, and requiring large expenditures of money."⁹

Intrinsic Value of Gas

Natural gas in the ground, or even in the field, has little intrinsic value in or of itself. The surface owner cannot ordinarily take the gas to market. The gas becomes of value primarily only as service is performed on it in delivering the gas under satisfactory working conditions to the ultimate consumer.

The right vested in the surface owner to remove the gas is always a property right and in many cases a valuable one. However, the gas

⁸ Frederick F. Sharpless: *Trans.*, vol. xlviii, p. 388 (1914).

⁹ Supreme Court of Appeals, W. Va., *Garrett vs. Oil Co.*, 66 W. Va., p. 591.

itself does not become property until reduced to possession, as explained later in this paper, under *Property Rights in Natural Gas*.

The value of the gas cannot be considered by itself, but must be considered with the leaseholds, wells, gathering lines, compressors, main lines, and distributing systems, all of which must be properly coordinated to give the gas in the field any value whatsoever. Furthermore, in considering the value it would not be policy to consider a single individual case, but the property as a whole must be taken together.

Quality

The quality considerations that affect the value of a natural-gas leasehold are as follows:

(a) Heating value.

(b) Purity, with regard to sulphur, moisture, and oil.

In this connection it is important to bear in mind that the quality is fixed by nature, and that it is economically feasible to alter this. On Apr. 13, 1915, the District Court of Shawnee County, Kansas, in the unreported case of *Ely vs. Public Utilities Commission*, No. 29,229, set aside the order of the Commission fixing a minimum heating value for natural gas. The decision has not been appealed.

Limits of Underground Reservoirs

There is absolutely nothing fixed from the surface, and while surface conditions may be indicative, the question of underground location can be established by the drill alone. Even the presence of gas sand is not necessarily an indication of the presence of gas.

"We judicially know, as a matter of common knowledge, that gas or oil does not exist in paying quantities under all lands within the recognized district, and there is no other generally acknowledged way than putting down a well to determine whether or not it does exist."¹⁰

Rock Pressure

The rock pressure of individual wells is not necessarily indicative of the amount of gas that may be obtained from such wells.

Criterion of Actual Flow

The open flow capacity of a well may be far from indicating the value of the well under routine operating conditions, since the actual line flow will always be less, and in many cases very much less than the open flow. If a well is located in a part of the field where it is not feasible to maintain

¹⁰ Supreme Court of Indiana, *Consumer's Gas Trust Co. vs. Littler*, 162 Ind., p. 326.

low line pressure into which the well may discharge, obviously the flow will be less than if the well were located where low line pressure might be continuously maintained.

Migratory and Fugitive Nature of Natural Gas

"Plaintiff assumes that there is a certain fixed amount of oil and gas under his farm, in which he has an absolute property. True, they belong to him while they are a part of his land; but when they migrate to the land of his neighbor, or become under his control, they belong to the neighbor."¹¹

"Oil and gas . . . have no fixed situs under a particular portion of the earth's surface within the area where they obtain. They have the power, as it were, of self-transmission. No one owner of the surface of the earth, within the area beneath which oil and gas move can exercise his right to extract from the common reservoir, in which the supply is held, without, to an extent, diminishing the source of supply as to which the other owners of the surface must exercise their right."¹²

"Oil and gas . . . unlike coal, iron and other minerals, they do not have a fixed situs under a particular portion of the surface, but are capable of flowing from place to place, and of being drawn off by wells penetrating their natural reservoir at any point. They are a part of the land, and belong to the owner so long as they are in it, or are subject to his control; but when they flow elsewhere, and are brought within the control of another by being drawn off by wells drilled in other land, the title of the former owner is gone. So, also, when one owner of the surface overlying the common reservoir exercised his right to extract them, the supply as to which other owners of the surface must exercise their rights, if at all, is proportionately diminished."¹³

Possession of Natural Gas

"The owner of the surface has no property right in the gas or oil until he has actually reduced it to possession, or, if he has any property right therein, it is a right in common with the co-equal right of other land owners to take from the common source of supply, and therefore subject to the legislative power to prevent a destruction of the common property of one of the common owners."¹⁴

"The property of the owner of lands in oil and gas is not absolute until it is actually in his grasp and brought to the surface."

"If possession of the land is not necessarily possession of the oil and gas, is there any reason why an oil and gas operator should not be permitted to adopt any and all appliances known to the trade to make the production of his wells as large as possible."¹⁵

"It is no longer an open question in this state that natural gas, when reduced to possession, becomes private property, and is a commercial commodity, which the owner may dispose of in whatever manner he may consider most advantageous."¹⁶

¹¹ Supreme Court of Pa., *Jones vs. Forest Oil Co.*, 194 Pa., p. 379.

¹² U. S. Supreme Court, *Ohio Oil Co. vs. Indiana*, 177 U. S., p. 190.

¹³ U. S. Circuit Court of Appeals, *Brewster vs. Lanyon Zinc Co.*, 140 Fed. Rep., p. 801.

¹⁴ U. S. Supreme Court, *Oil Oil Co. vs. Indiana*, 177 U. S., p. 190.

¹⁵ U. S. Supreme Court, *Oil Oil Co. vs. Indiana*, 177 U. S., p. 190.

¹⁶ Appellate Court of Indiana, *Richmond Nat. Gas Co. vs. Enterprise Nat. Gas Co.*, 66 N. E. Rep., p. 782.

Property Rights in Natural Gas

"The rule first announced in the case of *Hale vs. Reed*, 15 B. Mon. (Ky.) 479,¹⁷ and followed by the courts of last resort in all the great oil- and gas-producing States save one, Indiana . . . is stated by Mr. Justice Shiras in *Brown vs. Spilman*, 155 U. S., 665, as follows:

'Petroleum gas and oil belong to the owners of the land, and are a part of it so long as they are on it or in it, or subject to his control; but when they escape and go into other land, or come under another's control, the title of the former owner is gone. If an adjoining owner drills his own land and taps a deposit of oil or gas extending under his neighbors field, so that it comes into his well, it becomes his property.'

"The rule adopted by the courts of Indiana is this: The owner of the fee of oil- or gas-bearing lands does not have an absolute ownership in the oil or gas in place in the land, but a qualified ownership only, capable, however, of being made absolute by reduction to possession."¹⁸

"It must be held he who by lawful right reduces to his possession mineral, gas or oil, has the same absolute right of property therein, with the same power of barter, sale or other disposition, including, of necessity, the right of transportation and delivery under such reasonable rules and safeguards as the exigencies of the case may demand and the State employ, as the farmer has of his corn, his wheat, or his stock, or the merchant of his ware, and such absolute right therein as the State cannot deny him without just compensation, and any attempt to do so would be in violation of the fourteenth amendment to the federal constitution."¹⁹

"Gas and oil were likened to, not made identical with animals *feræ naturæ*, and, like such animals, were subject to appropriation by the owners of the soil, but also like them did not become property until reduced to actual possession.

"But an important distinction was pointed out. In things *feræ naturæ*, it was observed, all were endowed with the power of reducing them to possession and exclusive possession. In the case of natural gas, only the surface proprietors had such power, and the distinction, it was said, marked the difference in the extent of the State's control. In the one, as the public are the owners, everyone may be absolutely prevented from seeking to reduce to possession. No divesting of private property, under such a condition, can be conceived, because the public are the owners, and the enactment by the state of a law as to the public ownership is but the discharge of the governmental trust resting in the state as to property of that character. . . . On the other hand, as to gas and oil, the surface proprietors within the gas field all have the right to reduce to possession the gas and oil beneath. They could not be deprived of this right which belongs to them without a taking of private property."²⁰

"Natural gas after severance is a commodity which might be dealt in like other products of the earth, as coal and other minerals, and is a legitimate subject of interstate commerce; and that no State, by such laws as were involved in the case, can prohibit its transportation in interstate commerce beyond the lines of that State. The court held, after considering and construing the provisions of the act of 1907, that it was, upon its face, a law undertaking to prohibit the transmission or transportation in interstate commerce of natural gas to points beyond the State; that it was

¹⁷ Decided in 1854.

¹⁸ U. S. Circuit Court, *Kansas Natural Gas Co. vs. Haskell et al.*, 172 Fed. Rep., 545.

¹⁹ *Idem.*

²⁰ U. S. Supreme Court, *West vs. Kansas Natural Gas Co.*, 221 U. S., 229.

an unconstitutional interference with the rights of the complainants, who were legitimately engaged in that commerce, and that therefore the act was null and void."²¹

Natural Gas Should be Considered Only as a Utility Service

The only criterion of value that ought to be applied to gas leasehold valuation problems, is to consider the natural gas as an integral part of a public utility service.

Manufacturing Use of Natural Gas Should Not be Considered

Natural gas, although an ideal fuel for domestic service and manufacturing work, when used for industrial purposes must be sold at an absurdly low price in order to compete with coal and producer gas for industrial service. Although over 65 per cent. of the natural gas used at the present time is used for industrial service, this industrial use has had the inevitable effect of curtailing very materially the supply for domestic consumption.

Domestic Consumers Should be Given First Preference

On account of the exceptional value of natural gas for domestic service, every effort ought to be made to conserve the supply, so as to guarantee an adequate future, as well as an adequate present continuous service. It is doubtful whether anything does more for the prosperity, convenience, comfort, and in many cases the health of a community, than the introduction and continued use of natural gas.

Effect of Governmental Opposition to Unified Control

The present governmental tendency to prevent monopoly in the gas field, causes a decrease in the field of the leaseholds, and increases the cost of the gas to the public. Gas-field operating conditions should be regarded as a natural monopoly, so that in the development of the field one company could space the wells properly, and drain the field only to its safe working capacity, thereby greatly increasing and strengthening the life of the field.

Results of Competition Always Economic Waste

Competition in a gas field always results in a duplication of lines, unnecessary wells, enhanced operating cost, lack of proper coördination, failure to remove all the gas, and shortened life of the field, with the inevitable resulting injury to the domestic consumer.

²¹ U. S. Supreme Court, *Haskell vs. Kansas Natural Gas Co.*, 224 U. S., 217.

Ease in Drilling Stimulates Competition

The easier it is to drill a well in any given territory, the more wells will be drilled by small and inexperienced operators, and the greater will be the inefficient operation of the field. Furthermore, the indiscriminate drilling by inexperienced local operators always tends to increase the use of gas for manufacturing purposes, and takes the gas out at the fastest possible rate, thereby decreasing the effective life of the pool.

Drainage of Field

Under competitive conditions, even where the underground gas reservoir is made up of many local pools, various operators will drill into the same local pool, and thus drain out the gas from under each other's leaseholds.

Effect of Competition on Value

As the competitive conditions decrease the life of the field, and in other ways affect its stability, competitive conditions obviously decrease the value of the leasehold.

Effect of Competition on Cost of Gas

While competition decreases the value of the leaseholds, it increases the property value necessary in marketing the gas, by virtue of the large amount of development and inefficient operation, so that in the end the ultimate consumer pays more for his gas on a competitive basis than he would under conditions of regulated unified control.

Ethical Concept of Property

"The question of property is central and pivotal in modern distribution. Private property is the nourisher of mankind, the incentive to industry, and is the cement of society; it binds men together and is society's surest and firmest bond."²²

"The right to property is found in nature, sustained by organized society, and protected by the sanctions of the divine law. This right has its origin in a prior fact that each human being is a distinct individuality."²³

"The right of acquiring and possessing property and having it protected, is one of the natural, inherent and inalienable rights of man. Men have a sense of property, property is necessary to their subsistence, and correspondent to their natural wants and desires. Its security was one of the objects that induced them to unite in society. No man would become a member of a community in which he could not enjoy the fruits of his own labor and industry and the preservation of property is the primary object of the social compact."²⁴

²² Richard T. Ely: *Property and Contract in Their Relations to the Distribution of Wealth*.

²³ J. P. Newman: *Supremacy of Law*.

²⁴ U. S. Supreme Court, *Van Horne vs. Dorrance*, 2d Dallas, p. 304 (1795).

Ownership

In the United States we have been trained to regard property as a very holy thing. Here, property in the modern sense represents the basis upon which our social order was established. One of the most cherished clauses in our Constitution declares that private property shall not be taken for public use without fair compensation.

"The power to demand rent on land does not depend upon the manner of obtaining it, but upon the possession alone. Modern economic society does not ask a property owner how he became possessed of his property. The fact of possession is sufficient to yield him an income. Our civilization has been erected upon the theory of the validity of effort. Reward provides the stimulus of effort. It is reward, or the hope of reward that inspires. Deny the reward of effort, and the well-spring of effort is dried up."²⁶

State is Basis of All Ownership

"There is no such thing as natural property. Property is entirely the work of law. The idea of property consists in an established expectation; in the persuasion of being able to draw such and such an advantage from the thing possessed, according to the nature of the case. Property and law are born together, and die together. Before laws were made there was no property. Take away law and property ceases."²⁶

"Property implies the assent of the State, and in this we revert to the distinction between property and merely possession. If you have possession, when you lease the field another comes in and takes possession; if you have property, then the third person, the State, keeps out others although you be absent yourself. We cannot have property without law, for through law possession ripens into property."²⁷

Natural Resources Starting Point of All Value

The fact that natural gas was made by nature has been responsible for many erroneous ideas regarding its worth. Few people appreciate that all raw material is natural, and is the gift of God to man, and that "all production is carried forward upon the resources of nature by labor with the aid of capital. Every product of industry owes its origin to natural resources, the fields, the mountains, the water, some natural agent was the starting point for each material good on its way through the intricacies of the industrial system. Food, clothing, wealth in all its forms, is derived originally from nature.

"The forces of nature, working through the ages, have created things which mankind needs. Human effort expended on these products of nature, converts them into forms that are usable. All usable wealth, no matter what its form, owes its value in the beginning to nature's gifts, and after that the process of production."²⁸

²⁶ Scott Nearing: *Income*.

²⁶ Jeremy Bentham: *Theory of Legislation*.

²⁷ Richard T. Ely: *Property and Contract in Their Relations to the Distribution of Wealth*.

²⁸ Scott Nearing: *Income*.

Present Fair Value is True Basis

The same legal doctrines that are used in valuing other utility properties ought to be applied to natural-gas leaseholds. A long line of court decisions, from the United States Supreme Court down, establishes beyond question the doctrine that "present fair value" is the true basis. The crux of the whole matter is very aptly stated in the following typical water rate case:

"The value to be considered is the present fair value at the time the rate is fixed. The original cost is not at all conclusive, if it can be shown that it now has a different value, although the original cost is, as in all cases, an element which may be considered. The present fair value should be determined by the best evidence of which the nature of the case is susceptible. It should be measured by the fair market value of a similar right in the locality, or a similar locality, if such can be established by satisfactory evidence."²⁹

Real Effect of Leasehold Operation

"The removal of mineral substances from the land is an act which constitutes a permanent destruction of the substance of the real estate. It is a use of an estate which, unlike the use of a house or farm, consumes the things used. It no longer exists. It is obvious, therefore, that when one grants to another the right to thus exhaust the substance of a mineral estate, the exercise of the right, so far as it goes, works upon the estate, the same result, irrespective of the form of the instrument conferring such right. Where such instrument is in the form of an absolute conveyance of the mineral estate, little need be said. It becomes a simple case of a grant of real property by deed, requiring the usual formalities incident to such grant, and the instrument itself will be construed in the same manner as ordinary conveyances of real property.

"Where the instrument is not in the form of an absolute conveyance, but where the language employed is sufficient to pass the entire estate in mineral to the grantee, it often operates as a sale of real estate, although it purports upon its face to have a different scope and purpose."³⁰

How Leases are to be Valued

When gas leases are valued they are to be valued as property within the meaning defined herein.

Cost and "Value" Distinguished

To grasp clearly the distinction between value and cost is one of the first fundamental principles of valuation work. Very little reflection is needed to convince one that a thing may be worth much more than it

²⁹ Idaho Supreme Court, *Murray vs. Public Utilities Commission*, 150 Pac. Rep., pp. 50 to 51.

³⁰ Lindley on Mines, 3d ed., p. 2129.

cost, or that it may be worth much less. It may even be worth or valued much more at the present time than it cost, or it may be worth or valued much less.

Time an Element in Values

"Prices and values center upon the satisfaction of human needs in the present moment. The tendency is to avoid delay, and reduce as far as possible the period which elapses between effort and satisfaction.

"The present moment is the basis for computing and comparing values. To every one the enjoyment of wealth in the present is valued more highly than at any future period. No one will undertake long-time investments unless he expects an increase in the value of his estate in a measure corresponding to the paralleling and accumulation of cost and interest."¹

Sale Value as a Basis of Value

"In case a business changes hands by sale, the sale value so determined is accepted as the value of the business. In this transaction the price paid for the assets must be entered as their value on the books. Such sales thus establish a new recorded or book value for these assets. The fact of the sale is the most convincing evidence of value, although even this is not final proof, for one or the other of the parties may have been deceived or handicapped in the transaction. Nor will a past sale be accepted as absolutely determining a future sale even for the same property; in fact it is usually the reason for desiring a different value. Sales of property or business have a profound effect upon proprietary accounts in that the former owner receives at once all his income and capital, and can balance his books and determine his net profit, while the purchase is saddled with a cost or investment which forms the opening entry in a similar account, and no future acts will serve to reduce this initial cost."²

Appraisal of Market Value

"It is unnecessary to quote authorities to show that, in estimating the market value of land, everything which gives it intrinsic value is a proper element for consideration; not only its present use but its capabilities are to be considered."³

"The owner is entitled to the value of the property taken; that is, what it fairly may be believed a purchaser in fair market conditions would have given for it and not what a tribunal at a later date may think a purchaser would have been wise to give."⁴

Reflection of Hazard in Value

"The sale of an undeveloped resource is predicated upon an advance estimate of its prospective value for purposes of development, and whatever the theory of division of its appraised value between the land owner and the operator, there will exist a certain element of risk which necessarily involves the possibility of disproportionate profit or loss to either party and which naturally will be discounted by both. We must recognize that the risk thus created becomes a legitimate if not absolutely necessary basis for an additional item of cost in operation, and therefore will tend to increase both cost and selling price; so that if at this point there is introduced the third party to all such transactions, namely, the ultimate consumer, we discover who pays the carrying charge of this risk. This added item of cost may appear in the preliminary

¹ H. H. Chapman: *Forest Valuation*.

² *Idem*.

³ United States Supreme Court, *Wetmore v. Rymer*, 169 U. S., p. 128 (1897).

⁴ United States Supreme Court, *City of New York v. Sage*, 239 U. S., p. 57 (1915).

financing in the form of a larger interest rate offered to the bondholder, or of a larger discount given to the underwriter, or of a larger dividend promised to the stockholder, or in all of these combined."³⁵

Public is Served Best when Natural Gas Mining is Profitable

Natural gas has a heating value of from 50 per cent. to 100 per cent. greater than manufactured gas, and is also non-poisonous as distinguished from the poisons in all manufactured gas, and is therefore very much safer, especially for domestic use. Natural gas can do everything that manufactured gas need do, and many things that manufactured gas cannot do. In brief, man with all his skill has never been able to make a gas equal to that supplied by nature. Natural gas has usually been sold at prices far below the prices prevailing for manufactured gas, even without considering the increased worth of natural gas, due to its purity and much greater heating value, as against all manufactured gases. Therefore, every foot of gas that is found and served to the public represents a distinct economic gain to the community.

Natural gas can be found only by diligent prospecting. After it is found the service can be maintained continuously only by further continued development and persistent hunting for new supplies. In this development the prospector must figure on an average chance of getting one dry hole in every four wells drilled. Since the hazards are greater than in any other mining enterprise, the profits ought to be correspondingly greater. This element of profit is the only incentive which impels men to engage in so speculative an enterprise. If, in the aggregate, this amount of profit does not measure up to the hazards in the business the men will cease their work of prospecting and put their capital in safer enterprises. Wherever a close connection exists between effort and profit a strong resulting incentive is furnished for a further and continuous expenditure of effort. Therefore, a high rate of profit which will induce men to prospect continuously for natural gas brings about the condition that more people can use gas, and represents a distinct saving to the community.

Valuation Not an Exact Science

Some of the problems of valuation are unsolvable on any definite mathematic basis. To solve the problem as to what is the value of property requires the most logical reasoning, involving legal, economic and technical science, and in addition to all, that very rare endowment known as common sense.

"The ascertainment of value is not controlled by artificial rules. It is not a matter of formulas, but there must be a reasonable judgment having its basis in a proper consideration of all relevant facts."³⁶

³⁵ George Otis Smith: *Trans.*, vol. xlviii, p. 431 (1914).

³⁶ United States Supreme Court, *Minnesota Rate Case*, 230 U. S. 352.

Reproduction Method not Applicable

On account of the fugitive and migratory nature of natural gas, and since there is no regeneration, it is not possible to attempt to apply the reproduction method of valuation because it cannot be reasonably applied.

"The cost of reproduction method is of service in ascertaining the present value of the plant when it is reasonably applied and when the cost of reproducing the property may be ascertained with a proper degree of certainty, but it does not justify the acceptance of results which depend upon mere conjecture."²⁷

Classification

The well-known classification of leaseholds into producing, protective, reserve, and prospective groups, will answer all practical valuation problems. Producing leases are those on which producing wells have been drilled and are maintained. Protective leases are those which are contiguous to producing leases and which are held merely to protect the wells on the producing leases. Reserve leases are those which are held—usually after a certain amount of testing—as a reserve acreage to replace the depleting supply in the present producing wells. Prospective leases—sometimes called "wild-cat"—are those which have not been tested or proven, but are held because the indications are that they may contain gas.

Unit Prices

In fixing the value of natural gas leaseholds it is not possible to establish a market price, as is the case in ordinary commercial commodities which are constantly changing ownership in the open market. The unit price per acre to be applied to the different classes of leases must be fixed by expert testimony—that is, by the judgment of men skilled and experienced in the natural-gas business—in accordance with the principles herein defined, as so well stated in the following water right case:

"If no market value can be established, then the opinion of competent witnesses as to the actual value may be considered. In this respect the case does not present any exceptional features. The same rule is applied in the case of any property, real or personal. The fair market value is the usual standard; but if it be shown that the property has no market value, then witnesses may testify to actual value, which is, of course, largely a matter of opinion. Because it is difficult to determine the exact value of a certain kind of property, it does not follow that the owner shall be refused the protection of the law. The fair present value of the right is the ultimate fact to be found and considered. Exactly what probative or evidentiary facts shall be considered, or what standard of measurement shall be adopted in finding that ultimate fact, will depend largely upon the facts of each case as it arises."²⁸

²⁷ United States Supreme Court, Minnesota Rate Case, 230 U. S., 352.

²⁸ Idaho Supreme Court, Murray vs. Public Utilities Commission, 150 Pac. Rep., pp. 50 to 51.



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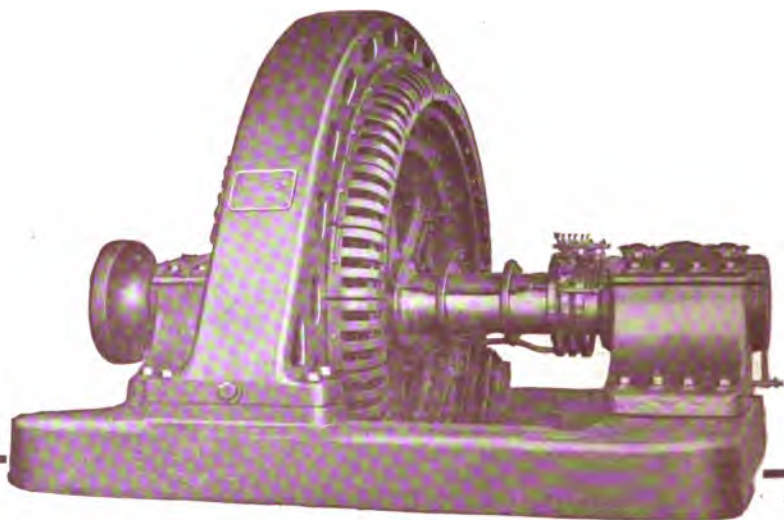
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